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MINERALS, METALS AND MINING TECHNOLOGIES

Mineral Processing - Research and Development

Compiled and edited by

L.L. Sirois, Mineral Sciences Laboratories

and

R.J.C. MacDonald, Office of Technology Transfer

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PREFACE

Mineral Processing - Research and Development is the first of a series of three separate volumes under the unified title "Minerals, Metals and Mining Technologies".

It comprises six state-of-the-art papers prepared by CANMET scientists involved in mineral research in Canada. Each paper provides a comprehensive review of mineral processing related to technology and research and development. The material is presented on a commodity basis under six main groupings: (1) Base metals/precious metals; (2) Iron and steel; (3) Industrial minerals; (4) Light and reactive metals; (5) Uranium; (6) Coal.

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FOREWORD

Canadian mineral and fuel production in 1983 amounted to $36 billion accounting for 9.3% of Canada's Gross National Product. The mineral industry in Canada provided jobs for an estimated 97 700 persons representing 1.1% of the Canadian work force. These figures exclude the petroleum and gas sectors which provided employment for approximately 52 600 persons.

In the field of mineral processing CANMET's objectives, as defined by Energy, Mines and Resources Canada, are:

(1) To ensure the availability to Canada of adequate technology for the mining, processing, and conservation of mineral resources.

(2) To increase the recovery and/or value of minerals from the processing of Canadian ores through the development of more efficient beneficiation, extraction and refining processes while minimizing energy use and environmental impact.

The purpose in publishing this volume, which was assembled by a number of authors over the period 1981-1983, is to provide an indepth review of the state-of-the-art of mineral processing in Canada in terms of general description of the industry, current technology, problems encountered, current Research and Development, and gaps and challenges.

W.G. Jeffery,
Director-General

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Base Metals/Precious Metals - L. Sirois, W. Craigen, M. Cristovici,
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Industrial Minerals - R. Collings, G. Riley, M. Malhotra,
K. Bell and P. Andrews
Light and Reactive Metals - C. Hamer and J. Skeaff
Uranium - G. Ritcey
Coal - T.D. Brown, L.L. Sirois, H. Hamza,
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INTRODUCTION

Mineral processing is primarily a manufacturing process for which ore is the raw material, and the derived product is that which best satisfies the demand for the primary consumer. Table 1 gives an overview of Canadian mineral production including the dollar value and Canada's position relative to other countries where possible. Three main branches of technology are involved in the production of an intermediate or finished product: ore dressing, which comprises the methods of separation of solid ores by means which do not effect substantial chemical change; hydrometallurgy which utilizes chemical reaction for separation of the constituents of solid ores or products; and pyrometallurgy which is involved in winning and refining metals from ores and concentrates, where heat is used.

Table 1 - Canada's mineral production (1980)(1)

	Unit measure (000)	Quantity	$ Value (000)	Rank (world)
Metals				
Aluminum	t	1 068	N/A	4
Antimony	kg	79	7 059	
Bismuth	kg	149	1 015	
Cadmium	kg	1 033	7 568	
Calcium	kg	531	3 422	
Cobalt	kg	2 118	134 748	6
Copper	kg	716 363	1 859 637	4
Gold	g	50 620	1 165 417	3
Iron ore	t	49 068	1 700 915	6
Iron (remelt)	t	N/A	118 990	
Lead	kg	251 627	273 766	4
Magnesium	kg	9 252	27 822	
Molybdenum	kg	11 889	299 323	3
Nickel	kg	184 802	1 497 418	1
Niobium (columbium)	kg	2 463	15 302	
Platinum group	g	12 776	159 088	3
Selenium	kg	279	8 779	
Silver	kg	1 070	828 805	4
Tantalum	kg	115	21 810	
Tellurium	kg	15	754	
Tin	kg	243	5 090	
Tungsten	kg	4 007	N/A	
Uranium	kg	6 739	702 038	
Zinc	kg	883 697	858 190	1
Total metals			9 696 956	

Table 1 (Cont'd)

	Unit measure (000)	Quantity	$ Value (000)	Rank (world)
Nonmetals				
Arsenious oxide	kg	N/A	N/A	
Asbestos	t	1 323	618 493	2
Barite	t	N/A	4 380	
Diatomite	t	N/A	N/A	
Gemstones	kg		1 632	
Gypsum	t	7 336	39 539	
Magnesium, dolomite, brucite	t		10 404	
Mica	kg	N/A	N/A	
Nepheline syenite	t	600	15 936	
Peat	t	32	345	
Potash	t	7 201	1 020 705	2
Pyrite and pyrrhotite	t	32	345	
Quartz	t	2 252	29 184	
Salt	t	7 423	122 775	
Soapstone, talc, pyrophylite	t	92	3 877	
Sodium sulphate	t	481	30 468	
Sulphur (in smelter gas)	t	895	28 913	
Sulphur (elemental)	t	7 656	444 095	
Titanium dioxide, etc.	t	N/A	117 060	
Total nonmetals			2 532 361	
Structural materials				
Clay products	t	N/A	108 453	
Cement	t	10 274	581 372	
Lime	t	2 554	129 232	
Sand and gravel	t	276 452	508 364	
Stone	t	103 366	341 156	
Total structural materials			1 668 577	
Fuels				
Coal	t	36 688	932 000	
Natural gas	000 m^3	87 108	6 148 826	
Natural gas byproducts	m^3	19 108	1 825 148	
Petroleum crude	m^3	83 477	9 037 890	
Total fuels			17 943 864	

Mineral processing is a complex series of events which are based on a number of unit operations chosen to suit the ore being treated and the product desired. Table 2 illustrates a number of the principal unit operations. Each of these can again be subdivided to fit requirements of ore and product. A choice of operations from preparation, beneficiation/concentration and extraction will produce a flowsheet of treatment from ore to metal.

Table 2 - Principal unit operations involved in mineral processing

Ore/material preparation	Ore concentration beneficiation	Metal extraction
Crushing	Heavy media separation	Roasting
Grinding	Gravity separation	Calcining
Classification	Electrical separation	Sintering
Sizing	Magnetic separation	Smelting
Sorting	Flotation	Converting
Waste disposal	Flocculation	Refining
	Dewatering	Vapormetallurgy
	Drying	Leaching
	Pelletizing	Cementation
	Tailings disposal	Solvent extraction
		Ion exchange
		Electrowinning
		Liquid/solid separation
		Slags, gases and tailings disposal

In the following background papers, the material is organized on a commodity basis under six main groupings (2):

1. Base metals/precious metals
2. Iron and steel
3. Industrial minerals
4. Light and reactive metals
5. Uranium
6. Coal

Each is discussed according to the following:

1. General description of the industry
2. Current technology
3. Problems
4. Current Research and Development
5. Gaps and challenges

The specific gaps and challenges emanating from the analysis of the six commodity groupings are numerous and complex and affect more than one grouping at a time. Generally, the gaps and challenges as discussed are related to the following issues:

1. The demand for improved working conditions in the mineral industry to reduce the risk of accident and health problems.
2. The demand that mineral-based activities reduce to a minimum potential damage to the natural environment.
3. The expanding mineral development in the less developed countries that will make it increasingly difficult for Canadian operators to remain competitive in world markets.
4. The continuing demand for self-sufficiency in mineral products such as chromium, manganese and other metals not produced in Canada.
5. The rapid development of the high technology industries such as computers, space, communications and new energy sources that will increase the demand for more exotic mineral-based materials.
6. The demand to conserve mineral resources through increased recovery of primary products and byproducts and by recycling.
7. The increasing cost of energy will force the industry to seek methods to reduce the energy component of various operations.
8. The current move from oil to electricity as an energy source will have increasing impact on the mineral industry in a number of ways.

9. The continuing reluctance of the Canadian mineral industry to diversify significantly into other areas such as the development and sale of machinery and other technology.
10. The increasing reluctance of people to work in remote areas where Canadian mineral activities usually take place.

A directory of federally supported research in universities from 1971/72 to 1980/81 is provided as an appendix after each of the main groupings except iron, steel and coal.

REFERENCES

1. "Canadian mineral survey - 1981"; Energy, Mines and Resources Canada, Can Min J; Feb. 1982.

2. Gow, W.A. "Mineral processing background paper"; Memorandum, July 8, 1982.

BASE METALS/PRECIOUS METALS

GENERAL DESCRIPTION OF INDUSTRY

In Canada as of January 1981, 100 mines or 36% of all operating mines produced precious metals or sulphide ores of one kind or another (1). Table 3 illustrates the scope and magnitude of this portion of the mineral industry.

The metal complexity of the sulphide ores varies from the single species low grade porphyry copper, to multicomponent nickel-copper, to fine grained complex zinc-lead-copper, with most of them including important byproducts. Of the precious metals, 100% of the platinum, 80% of the silver and 30% of the gold are recovered as byproducts from sulphides. The mines vary in size from 100 to 45 000 t/d and are distributed throughout Canada.

About one half of Canada's nonferrous mine production leaves the country in concentrate and relatively unprocessed forms (3). The balance is treated in Canada in seven copper smelters and three refineries, four nickel smelters and three refineries, two lead smelter/refineries and four zinc smelter/refineries. In the recovery of the byproducts and elimination of impurities such as arsenic and mercury in an environmentally acceptable manner, the major problem facing Canada's smelters in the future is undoubtedly the control of SO_2 emissions and its effect on Canada's competitive position in world markets.

Table 3 - 1980 Canadian production of sulphides, precious metals and byproducts (2)

Metal	Unit measure	Quantity (000)	$ Value (millions)	World ranking
Copper	t	716.4	1 859.6	4
Lead	t	251.6	273.8	4
Zinc	t	883.7	850.2	1
Nickel	t	184.8	1 497.4	1
Molybdenum	t	11.9	299.3	3
Cobalt	kg	211.8	134.7	6
Platinum group	g	12 776.0	159.1	3
Gold	g	50 620.0	1 165.4	3
Silver	g	1 070.0	828.8	4

CURRENT TECHNOLOGY

BENEFICIATION

Beneficiation is the first step in recovering the contained metals from sulphide ores. Its purpose is to produce a concentrate for the extraction processes by reducing the content of gangue and other unmarketable material to the extent possible. Typically, it consists of two major operations - comminution which includes crushing and grinding, and a concentrating process which, with base metal sulphide ores, usually takes the form of flotation. Figure 1 describes a generalized flowsheet for the concentration of base metal sulphide ores (4).

The mineralogy of the ore determines the exact technology of the milling method (5). Of particular importance is grain size and degree of intergrowth of valuable minerals with each other and the gangue minerals. The very coarse run of mine material is first reduced by jaw or gyratory crushers and then fed to rod and ball mills to attain the liberation size required by the ore. Depending on the competency of the ore, single stage autogeneous milling may replace crushing and grinding. The comminution stage is usually the most expensive unit operation in any ore treatment as it is very energy intensive. The finer a material has to be ground the higher the cost. No new major technological breakthroughs in comminution have surfaced to greatly reduce costs except by making the machines stronger and larger.

All Canadian base metal sulphide ores are concentrated by flotation. Flotation is a unit operation whereby the use of specialized chemicals absorbing on the surface of selected minerals permits them to become attached to air bubbles generated in flotation cells. The bubbles then float to the top in the form of a froth thus separating themselves from the non-float material which is

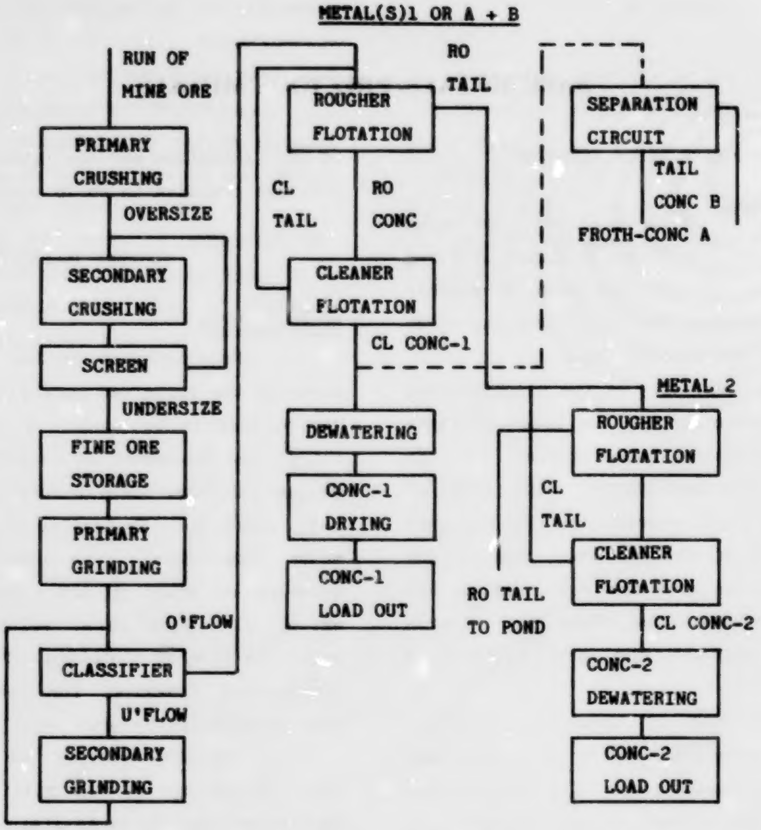

METAL(S)1 OR A + B
RUN OF MINE ORE
PRIMARY CRUSHING
OVERSIZE
SECONDARY CRUSHING
SCREEN
UNDERSIZE
FINE ORE STORAGE
PRIMARY GRINDING
O'FLOW
CLASSIFIER
U'FLOW
SECONDARY GRINDING
ROUGHER FLOTATION
RO TAIL
CL TAIL
RO CONC
CLEANER FLOTATION
CL CONC-1
DEWATERING
CONC-1 DRYING
CONC-1 LOAD OUT
SEPARATION CIRCUIT
TAIL
CONC B
FROTH-CONC A
METAL 2
ROUGHER FLOTATION
CL TAIL
RO TAIL TO POND
CLEANER FLOTATION
CL CONC-2
CONC-2 DEWATERING
CONC-2 LOAD OUT

Fig. 1 – Generalized flowsheet for concentration of base metal sulphide ores

removed from the bottom of the cells usually as rejects or tailings. Flotation is governed by the science of surface chemistry which has not yet unlocked all the secrets affecting this unit operation.

For copper-zinc and lead-zinc ores, sequential selective flotation is employed with a copper or lead concentrate floated first. For copper-lead-zinc ores, the most efficient route is to float copper and lead minerals together in a bulk concentrate and then produce separate concentrates by selectively floating one mineral from the other by the use of selective depressants. Molybdenum is also recovered by flotation while for ores containing tin, the tin is recovered by gravity concentration from the flotation tailings.

For nickel-copper ores, the usual process is to float a nickel-copper concentrate and produce a pyrrhotite concentrate by magnetic separation ahead of the nickel-copper flotation or by flotation after the pentlandite-chalcopyrite collecting stage. The nickel-copper concentrate can be delivered as such to the smelter or subjected to selective flotation where the pentlandite is depressed and the chalcopyrite is floated. In this case the two products go to separate metallurgical treatment. Nickel in the copper-concentrate is not usually recovered in conventional copper smelting. The pyrrhotite product is reground and again subjected to a nickel-copper scalping flotation while depressing the pyrrhotite. The pyrrhotite can be cleaned further for iron recovery or stockpiled within the tailings disposal area. Cobalt, if present, can be recovered by gravity concentration or flotation. In Canada, cobalt is associated with the copper-nickel ores where it is recovered in the copper-nickel concentrates and separated during metallurgical treatment.

South Africa is the largest producer of platinum, where it occurs in platinum metals-bearing ores. It is recovered by gravity and flotation methods before shipment to smelter and refinery. In Canada, platinum is recovered as a by-product from the nickel-copper ores. The platinum bearing residues from copper and nickel refining are shipped to either England or Norway for further processing.

Native silver and silver minerals are readily recovered by flotation provided they are free and floatable. Generally, the greatest proportion of silver is recovered from lead and copper concentrates and a smaller amount from zinc concentrate (6). On straight silver ore a combination of gravity concentration and flotation is employed to concentrate the silver.

Cyanidation is employed universally to recover gold. All of the ore may be cyanided or if gold is associated with sulphides a gold-bearing sulphide concentrate is floated and cyanided - in most cases it is necessary to roast the flotation concentrates prior to cyanidation. After solids-liquid separation which consists of thickening, filtering and clarification, the gold is precipitated from the solution with zinc dust. The gold precipitate is filtered by filter presses and periodically collected and smelted with fluxes to produce a final impure gold bar product. An alternative technology for gold recovery is the carbon-in-pulp cyanidation method. Its main advantage is in eliminating the sometimes difficult solids-liquid separation step. Also, for gold ore containing arsenic minerals it eliminates the possibility of toxic arsene formation. In this method, carbon is added to the agitators to absorb the gold as soon as it dissolves in the weak cyanide solution. Gold is then stripped from the loaded carbon by elution with a hot caustic cyanide solution. The gold is extracted from the eluate by electrowinning on a steel wool cathode while the stripped carbon is recycled for re-use. Carbon-in-pulp cyanidation will be employed at the Detour Lake mill which is under construction. In recent years a number of cyanide heap leaching operations have started in the southwest USA. This method has proven to be economically feasible for gold recovery from low-grade ores provided the cyanide solution can percolate through the ore.

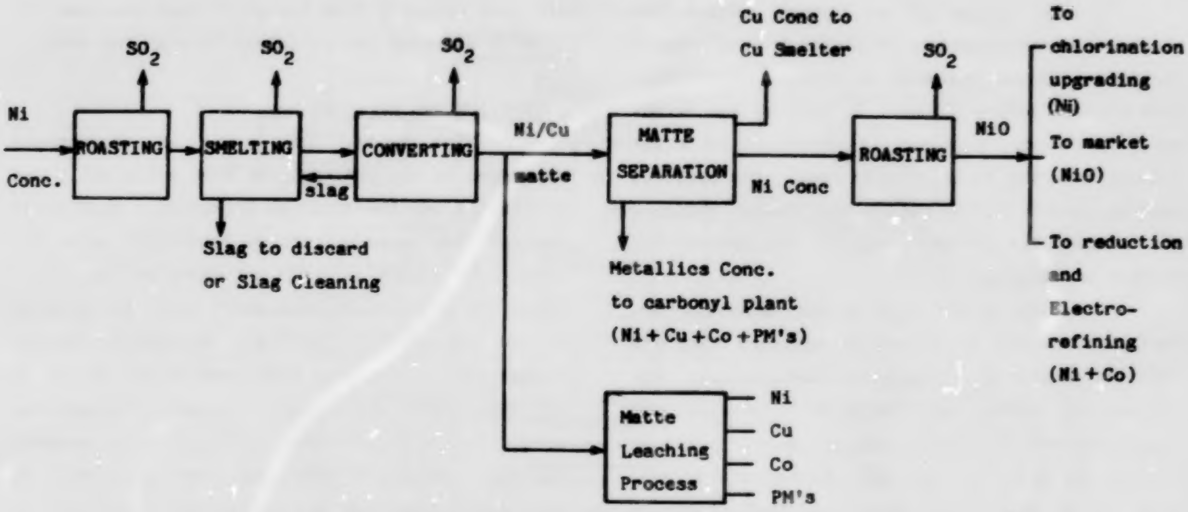
EXTRACTION AND REFINING

The better the grade of concentrate delivered to the smelter, the more efficiently and profitably can the metal be extracted. High grade concentrates usually involve trade-offs with the amount of usable minerals rejected to the tailings. In non-complex base metal ores, the problem is not particularly serious. Acceptable concentrates can be produced with recoveries in excess of 90%. But with multicomponent fine-grained ores, the difficulties in segregating several desirable minerals from each other, as well as separating them from the waste, are such that, to produce concentrates leading to profitable smelting, overall recoveries can be less than 70%. The metal sulphide concentrates used as feed to the different smelters usually vary from 10% to 15% for nickel, 20% to 30% for copper, 50% to 55% for zinc and 30% to 60% for lead. Figure 2 gives an example of a simplified smelting/refining process for a nickel concentrate to demonstrate the complexity of base metal extraction.

The technologies employed in Canada's base metal smelting and refining industry vary from old to new. They not only differ in commodities, but the technologies can vary substantially for the same commodity. For simplification therefore the industry has been segmented on a major commodity basis for identifying and evaluating existing technologies. A detailed description of current technology of extraction and refining follows.

Copper Smelting and Refining (Pyrometallurgical Processes)

Canada has seven copper smelters at a total capacity of about 635 000 t/a of copper, 94% of which is in the form of impure blister copper. One company produces Ni/Cu matte which is further converted and refined in the company's overseas plant. In addition, Canada has three electrolytic copper refineries capable of refining all of the blister copper into high purity cathode and wire bar copper.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SO2
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Ni Conc
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To market (NiO)
To reduction and Electro-refining (Ni + Co)
Matte Leaching Process
Ni
Cu
Co
PM's

Fig. 2 - Simplified smelting/refining processes for nickel concentrates

No fewer than six copper smelting technologies are in commercial use throughout the world, five of which are in Canada.

Reverb smelters

Reverb smelters use air and carbonaceous fuel to produce low grade (40-45%) copper matte and discard iron bearing slag. The matte is then converted to blister copper by air blowing in batch converters to remove additional sulphur and slag from most of the remaining iron. The converter slag, which contains appreciable copper, is recycled to the smelter to minimize losses.

More than 40% of Canada's copper smelter output is produced from three smelters utilizing this technology. Two of the smelters (HBMS and Noranda) are very old and do not have acid plants, mainly because the technology is not amenable to SO_2 and conversion to acid, and also because their location makes sulphuric acid marketing very difficult. SO_2 is therefore vented to atmosphere. The third smelter (Gaspé) utilizes a fluid bed roaster prior to smelting to drive off about 40% of the sulphur in a continuous high strength SO_2 gas stream, which is then captured and converted to acid in an on-site acid plant. Some of the converter gases are also sent to the acid plant bringing total SO_2 capture to about 60%. The remaining 40% is vented to atmosphere as a dilute SO_2 gas stream.

There are two major problems with the conventional reverb smelt or roast/reverb smelt, batch conversion process:

1. The reverb gas streams are too dilute and the converter gas streams are too variable for efficient capture and conversion of SO_2 to H_2SO_4 in an acid plant.
2. The smelting process is very energy inefficient compared with modern copper smelting processes.

A method has been developed to modify the reverb furnace to use oxygen enriched air, thus greatly reducing fuel requirements and simultaneously producing SO_2 concentrations high enough for efficient conversion in an acid plant (7). This technology is being tested on a commercial scale in the USA. In Canada however, because of the age and location of the smelters, it is likely that complete replacement of the reverbs with one of the modern copper smelting processes will be the most economical long term solution. However, modification may be an economical short term solution if SO_2 controls have to be implemented in a relatively short term.

Flash smelters

The Inco flash smelter was commercialized in 1952, and the present version has operated in Sudbury since 1968 (8,9). A second Inco flash smelter is presently being built in the USA. Concentration is injected with pure oxygen in a "flash chamber" to produce 45% to 55% copper matte, discard slag and 70% to 80% SO_2 stream which is recovered and sold as liquid SO_2. Since the SO_2 concentrations are so high, gas handling, cleaning, and SO_2 recovery costs are minimal. The process is also exceptionally energy efficient, requiring essentially no carbonaceous fuel, except for predrying the concentrate.

The Outokumpu flash smelter is different in design from Inco's and uses preheated air or oxygen enriched air, rather than pure oxygen (8,10). Off-gases therefore contain lower SO_2 concentration and sulphuric acid is produced by a byproduct rather than liquid SO_2. The smelter is not as energy efficient as Inco's and requires a high recovery and utilization of waste heat from the smelter off-gases to minimize total energy requirements. However, cheap carbonaceous fuels (coals, heavy oils, etc.) can be used and fuel requirements are low at high oxygen enrichments. The Outokumpu smelter has the flexibility to produce copper mattes grading up to 70% copper. Although at these high matte grades the load in the converters is considerably reduced, the smelter slag is too high in copper to be discarded and must be processed for copper recovery, either in an electric furnace, or by slow cooling, milling and flotation.

There are no Outokumpu flash smelters in Canada. However, the more than 25 Outokumpu smelters throughout the world is a tribute to their aggressive marketing ability.

In both processes, conversion of the matte to blister copper is still carried out batch-wise in conventional Pierce Smith converters. Also, the converter slag, which is high in copper, is usually recycled to the smelter. In the Inco process, conversion results in 40% to 45% of the sulphur being released as a variable concentration, variable flow gas stream which is not readily amenable to recovery in an acid plant. In the Outokumpu process this can be reduced to 25% to 30% of the sulphur if 60% to 70% copper mattes are produced. In either, because of the batch nature of the process, control of fugitive emissions and SO_2 concentrations and flows is difficult and costly. What is needed therefore is a continuous conversion process which would facilitate SO_2 recovery.

Top-blown rotary converter (TBRC)

Afton Mines in British Columbia was the site of the first commercial application of a TBRC for copper extraction, in which both smelting and converting are carried out batch-wise in a single vessel (8,11,12). The process is ideally suited for Afton Mines concentrate, which is primarily native copper having a very low sulphur content. Pure oxygen, introduced through a water cooled lance, is used for smelting and converting. Oil or natural gas is introduced with the oxygen on an as required basis, with coal or coke also being used for slag cleaning and refining of the blister copper. The ability to adjust the TBRC rotational speed and lance angle facilitates high utilization of fuel and oxygen, thus it is reasonably energy efficient. However, its completely batch nature makes SO_2 recovery difficult and expensive. At Afton, SO_2 and volatile impurities are removed in a dual alkali scrubber. This is an economical approach only because of the very high copper and very low sulphur content of the concentrate. Conventional copper concentrates would require SO_2 recovery by conversion to acid, fertilizer or other saleable products. Although the tight-fitting water cooled hood minimizes air in-leakage and the use of oxygen can provide high SO_2 concentrations, the batch nature of the process makes it unlikely to be as attractive, particularly for SO_2 control, as alternative copper smelting technologies. The TBRC however is very flexible, and can be used for smelting, converting, slag cleaning or refining. It could therefore find other specialized applications for treating impure copper concentrates or residues or for conventional converting of matte to blister copper (13).

Continuous smelters/converters

The Noranda continuous reactor was commercialized in Canada in 1973 (8,14,15). A second installation commenced operation in the USA in 1978. The reactor was originally designed to produce blister copper directly, using air or oxygen enriched air and any carbonaceous fuel to provide the required additional energy. Today, for various reasons, some related to the control of impurity levels in the blister copper, the reactor operation is controlled to produce 75% copper matte using tuyere injected oxygen enriched air (up to 35%). At this level of enrichment, carbonaceous fuel requirements are low, and a continuous high strength SO_2 gas (more than 20%) is produced which is readily amenable to recovery in an acid plant, however, because of acid marketing problems, Noranda does not have an acid plant at the smelter.

Reactor slag is too high in copper for discard. Slag therefore is treated by slow cooling, milling and flotation to recover the copper for recycle to the process. Overall copper recovery in the Noranda process is exceptionally high.

The high grade copper matte is converted to blister copper in conventional Pierce Smith converters, and the converter slag is recycled to the reactor. The batch conversion step again results in about 20% of the sulphur being released as a variable SO_2 gas stream, not readily amenable for recovery. Although operating the reactor to produce blister copper would eliminate the batch converting operation, the lower reactor throughput and the problem of controlling impurities such as arsenic, bismuth and antimony, appear to negate the advantages of having all of the sulphur removed in a continuous high concentration SO_2 stream. As is the case with the flash smelters what is really required is a separate conversion furnace which can be operated continuously.

Another version of the Noranda reactor called the "Q-S oxygen process" has been proposed for producing blister copper and discard slag in a single reaction vessel (8,16). The process however requires a great deal more development to verify its technical feasibility and commercial viability.

The Mitsubishi continuous process was commercialized on a relatively small scale in Japan in 1974 (8,17,18). A larger commercial plant began operation in Canada at Kidd Creek mines in 1981. The process utilizes oxygen enriched air and is the only truly continuous commercial pyrometallurgical process for copper. Essentially all of the sulphur in the concentrate is released in a continuous stream of high concentration SO_2 gas (up to 30%) which is recovered in an acid plant. The process consists of a smelting furnace, an electric slag cleaning furnace and a converting furnace. Reaction gases are introduced through lances above the molten bath surface. Matte and slag are transported in heated launders between the furnaces. Matte grade is 65%. Slag is cleaned in the slag cleaning furnace prior to discard, while matte is converted to blister copper in the converting furnace. Converter slag is recycled to the smelting furnace. Gas streams from the smelting and converting furnace are combined for SO_2 recovery in the acid plant. Solid, liquid or gaseous carbonaceous fuels can be used in the process. Fuel consumption is lowest and throughput is highest at high oxygen enrichments.

Since the process is completely continuous it is excellent for efficient and inexpensive SO_2 recovery and therefore ideal from a environmental point of view. However, because of the heating requirements for the launders and the energy requirements for electric furnace slag cleaning, it is unlikely as energy efficient as the Inco or Noranda processes. Also some doubt exists about its ability to handle high impurity concentrates. This is probably the major area of concern for any continuous smelting/converting process, and is the main reason the batch converting is still preferred.

The KHD cyclone smelting continuous converting process was developed in Germany by Kloeckner-Humboldt-Deutz (8,19,20). The process utilizes a smelting/slag cleaning furnace and a converting/refining furnace. Smelting is carried out in a "cyclone smelter" using 100% oxygen. Smelting rates are very high and the cyclone smelter has unusual flexibility with respect to throughput rates and matte grade, which can be varied from 70% to 110% of rated capacity and 46% to 80% Cu respectively.

Slag from the cyclone smelter, containing 6% to 8% copper, is treated in a continuous top-blowing (contop) furnace using a reducing gas to produce discard slag containing less than 0.5% copper. Matte from the smelter flows through a closed launder to a continuous contop converting/refining furnace where oxygen and a reducing gas are used to produce low sulphur low oxygen blister copper. Converter slag is recycled to the smelter. Off-gases from the smelting/slag cleaning furnace and from the converting/refining furnace are combined to produce a greater than 40% SO_2 gas stream which is recovered in an acid plant.

The KHD process has been demonstrated on a pilot plant (1000 kg/h) scale in Germany but to date no commercial plant has been built. The process has many potential advantages, including high smelting rates and potentially low capital costs, which could make it attractive for relatively small throughput plants. The completely continuous nature of the process and the use of 100% oxygen for both smelting and converting make it ideal for efficient and inexpensive SO_2 recovery. The high oxygen utilization efficiency should result in reasonably low energy requirements for the process, and the relatively high operating temperatures should facilitate the removal of undesirable impurities. The technology therefore may come closest to meeting the ideal requirements for a pyrometallurgical copper process. Development work, however is far from complete and a great deal more information is required, particularly on the operation of contop furnaces, to verify the technical and economic viability of the technology.

Copper refining

Regardless of the smelting/converting process, blister copper is usually further treated in anode furnaces to reduce sulphur and oxygen levels prior to casting into anodes for electrorefining. In the refinery, well established electrorefining techniques, employing sulphuric acid/copper sulphate electrolyte, are used to produce high purity cathode copper, and to recover precious metals and other byproducts in the form of insoluble slimes, which are further refined to high purity products. Impurity build-up in the electrolyte is controlled by means of a bleed stream which is treated for copper and often nickel recovery prior to discard.

There are very few problems associated with present day copper refining processes. Some impurities such as arsenic, antimony, bismuth, selenium and silver can pose problems if levels in the blister copper are too high (21). It may be desirable therefore to investigate ways of minimizing these problems in the refining process to facilitate the development and use of continuous smelting/converting processes which are not constrained by impurity levels in the concentrate and/or the blister copper.

Hydrometallurgical Processes for Copper

A great many hydrometallurgical processes have been proposed and investigated for copper extraction from concentrates over the past 25 years. However, very few have been carried to a pilot plant scale and only one is presently operating on commercially. This is the clear process, developed and commercialized in the USA by Duval Corporation at 40 000 t/a (22). It consists of a cupric/ferric chloride atmospheric leach to dissolve the copper and iron from the chalcopyrite concentrate, while converting sulphides primarily to insoluble elemental sulphur. Copper is recovered by electrowinning from cuprous/ferrous chloride media in diaphragm cells, where the leachant is regenerated in the anolyte. Electrowinning from cuprous/ferrous solution in this manner minimizes power requirements. Copper is recovered from the cell as coarse crystalline metallic particles. Iron is rejected as discard hydroxide/jarosite by pressure oxidation with oxygen. Most impurities in the concentrate are also removed in the iron residues.

A similar process was developed and piloted in the USA by Cyprus Mines Ltd. (23). The main differences are that the copper is recovered by hydrogen reduction of selectively crystallised cuprous chloride and that the iron is rejected by oxydation at atmospheric pressure.

Other processes similar to those of Duval and Cyprus have been piloted in France and Norway (24,25). Two main problems with these are the difficulty in recovering precious metals, particularly gold and producing wire bar grade copper directly in the process. The latter problem is primarily due to the silver content of the copper product. Most of the silver in the concentrate leaches with the copper and a satisfactory method of separating and recovering the silver prior to copper recovery has not yet been developed.

Another somewhat different method, the S-C copper process, has been developed and piloted in Canada by Sherritt Gordon and Cominco (8,26). It employs an oxidation reduction roast to facilitate selective sulphuric acid dissolution of the iron in the copper concentrate. After solid/liquid separation, iron is precipitated from the leach liquors as jarosite for disposal to tailings, while copper in the leach residue is extracted by pressure oxygen leaching and recovery by conventional electrowinning from purified sulphate electrolyte. Precious metals and molybdenum remain in the pressure leach residues. Approximately 50% of the sulphur in the concentrate is converted to SO_2 and H_2S, which are converted to sulphuric acid and elemental sulphur (claus plant) respectively. Most of the remaining 50% is converted directly to elemental sulphur in the pressure leach.

Although the S-C process produces wire bar grade cathode copper it is not as energy efficient as the chloride leach processes because electrowinning is carried out from cupric solutions and the anode reaction generates oxygen. Power requirements therefore can be up to four times those required for cuprous chloride electrolysis in diaphragm cells. The capital costs of the process are also considerably higher because of the requirement for a roaster, an acid plant and a claus plant. Although precious metal and molybdenum recovery from the leach residues should be possible, the technology and associated costs for so doing have not been adequately demonstrated.

Another hydrometallurgical process for copper which has been investigated on a pilot scale is the Arbiter process (27). This process utilizes low pressure ammonia-ammonium sulphate leaching to solubilize copper and convert iron to an insoluble hydroxide. Sulphides are converted to ammonium sulphate, which is boiler with lime to recycle ammonia to the process and remove sulphate as gypsum. Copper is recovered by solvent extraction, followed by electrowinning from acid sulphate media. Precious metals remain in the leach residues and are said to be recoverable by cyanidation.

The process has never been commercialized probably because the high cost reagent (ammonia), high energy requirements (electrowinning, lime boil) and difficult solid-liquid separation have combined to make the process economically unattractive.

Nickel Smelting and Refining
(Pyrometallurgical Processes)

Canada has three nickel smelters and one hydrometallurgical nickel extraction plant, with a total rated capacity of about 265 000 t/a, approximately 80% of which is further processed in Canada to refined nickel products. One company produces Ni/Cu matte which is refined in the company's overseas plant.

Numerous nickel extraction/refining technologies, both pyrometallurgical and hydrometallurgical are in use or under development throughout the world. However, many are for recovering nickel and byproducts from laterite deposits or from sea nodules. Since neither of these are ever likely to become a source of nickel for Canadian smelters only those technologies that are applicable to sulphide nickel resources are described, whether they occur naturally, or are produced artificially as a method of recovering the nickel in a concentrated form from the other ore sources.

It should also be noted that nickel extraction/refining technologies are more complex than those for copper, primarily because copper, cobalt and platinum group metals are nearly always associated with nickel sulphide deposits, and efficient recovery of these byproducts has a profound effect on process economics. The high iron to nickel ratio of nickel concentrates also puts more stringent demands on slag cleaning techniques to maximize the recovery of nickel and associated byproducts in the pyrometallurgical extraction processes.

Reverb smelters

Inco's Copper Cliff nickel smelter was built in 1930 (28,29). Nickel concentrates containing about 10% Ni and 2% Cu are partially roasted with air in multi-hearth roasters and smelted in reverberatory furnaces to produce a 20% to 25% Ni/Cu matte and discard slag. The matte is further oxidized to remove iron and sulphur in conventional Pierce Smith converters. Converter slag is recycled to the reverb furnaces. Converter matte containing about 50% nickel, 26% copper and 0.5% iron is cast into moulds for slow cooling and separation by grinding, flotation and magnetic separation into a nickel sulphide concentrate (70% Ni, 1% Cu), a copper sulphide concentrate (73% Cu, 5% Ni) and a metallics concentrate (65% Ni, 20% Cu) which also contains most of the precious and platinum group metals in the original concentrate. The copper concentrate is converted to blister copper in the copper smelter, and the copper converted slag, containing residual nickel is recycled to the nickel smelter. The nickel concentrate is roasted with oxygen enriched air in fluid bed roasters to produce nickel oxide for direct sale, or for further refining by a chlorination hydrogen reduction process to remove copper and produce a 95% Ni alloy. Nickel oxide is also sent to the Port Colborne refinery for reduction melting with coke, casting and electrorefining to produce high purity nickel, and copper and cobalt byproducts.

The metallics concentrate is processed in the Copper Cliff nickel refinery (built in 1973) to produce high purity nickel pellets and powder and ferro-nickel powder (29,30). The process involves melting and controlled desulphurizing with oxygen in a TBRC. The molten metallic product is water granulated and fed to a high pressure moderate temperature carbonylation process, where nickel preferentially reacts with CO to form nickel carbonyl which is liquefied and further purified in a distillation column to remove iron. The nickel carbonyl is then thermally decomposed to form high purity nickel pellets and CO for recycle to the process. Iron in the metallics concentrate which has reacted to form iron carbonyl is recovered as ferro-nickel powder (29% Ni). Copper, residual nickel, cobalt and precious and platinum group metals are recovered from the carbonyl plant residues. The residues are oxygen pressure leached with spent electrolyte to selectively dissolve nickel and cobalt which are precipitated as carbonates for recovery and further refining to nickel and cobalt products. Copper is also recovered by oxygen pressure leaching, followed by purification and electrowinning to product high purity cathode copper. Precious and platinum group metals, which remain in the leach residues, are sent to Port Colborne and overseas for concentration and recovery.

As with copper reverb furnaces the major problems at the Copper Cliff nickel smelter are related to SO_2 emission control and energy efficiency. About 90% of the total sulphur is removed as SO_2; 50% from the roaster/reverb and 40% from the converters.

1. The multi-hearth roaster and reverb furnace gas streams are too dilute and the converter gas streams are too variable for efficient capture and conversion of SO_2 in an acid plant.
2. The smelting process is very energy inefficient and does not take advantage of the sulphur fuel in the concentrate to minimize carbonaceous fuel requirements.

All of the SO_2 presently produced in the nickel smelter is vented to atmosphere. To maximize SO_2 capture and minimize energy requirements a continuous smelting/converting process is desirable which can utilize oxygen to produce a high grade nickel/copper matte and a discard slag, while simultaneously producing a high strength SO_2 gas stream and minimizing carbonaceous fuel requirements. Pre-roasting of the high iron nickel concentrates may be necessary for energy balance considerations. This can be carried out in fluid bed roasters which will also provide a continuous high strength SO_2 gas stream.

Electric furnace smelters

Inco's Thompson Nickel Smelter and Refinery was built in 1961 (29,31,32). The smelter utilizes fluid bed roasters to partially roast the concentrate which is then smelted in an electric furnace to produce 25% nickel matte and a discard slag. The matte is further oxidized in conventional batch converters to remove iron and sulphur and produce 75% to 80% Ni/Cu matte which is cast into anodes for electrorefining. Converter slag is recycled to the electric furnace. In the refinery a chloride/sulphate electrolyte at controlled pH is used to produce high purity nickel cathodes from the impure matte anodes. A sophisticated electrolyte purification circuit is required to remove arsenic and iron, and recover cobalt and copper byproducts. In addition anode bags are required to contain the anode residues and prevent migration of impurities into the cathode compartment. The anode residues are treated for the recovery of precious metals and elemental sulphur, which is produced in the electrolysis reaction. The undissolved portion of the anode is recycled, partly to the smelter, and partly to the refining process to maintain the required nickel levels in the electrolyte.

The Falconbridge nickel smelter was completed in 1978 and is essentially a modern version of Inco's Thompson smelter, utilizing fluid bed roasters, electric furnaces and converters to produce a 75% Ni/Cu matte (33). The matte is shipped to Falconbridge's refinery in Norway, where it is treated to produce refined nickel, copper and cobalt products, and recover precious metal byproducts. Two refining processes are used. In the Falconbridge matte leach process the matte is leached in strong HCl solution to selectively dissolve the nickel (34). The solution is then purified by solvent extraction techniques to remove iron and recover cobalt and any copper that is leached. High purity nickel chloride is crystallized from solution and converted to nickel oxide by oxy-hydrolysis in a fluid bed reactor. High purity nickel granules are produced by hydrogen reduction of the NiO in a rotary kiln. Most of the copper and all of the precious and platinum group metals remain in the leach residue. Copper is recovered by roasting and sulphuric acid leaching, followed by conventional purification and electrowinning to produce cathode copper. Precious and platinum group metals are recovered from the acid leach residues.

In the matte leach process most of the sulphides are converted to H_2S which can be readily recovered as elemental sulphur in a claus plant, or oxidized to SO_2 for recovery as acid or liquid SO_2. The process has additional flexibility in that nickel can be recovered by direct hydrogen reduction of the chloride or by electrowinning from chloride solution (35). To date however the process has been operated only on a relatively small scale (7000 t/a Ni).

The second refining process utilizes two-stage controlled aqueous chlorine leaching to selectively leach the nickel and cobalt from the nickel/copper matte, while converting sulphides to elemental sulphur (36,37). Cobalt is separated by a solvent extraction technique and the nickel chloride solution is then purified to remove iron and lead. Nickel and cobalt are both recovered by electrowinning, and chlorine produced in the electrolysis cell is recycled to the process. All of the copper and precious metals, along with residual nickel remain in the leach residue. The copper is recovered by roasting, selective sulphuric acid leaching and electrowinning. Residual nickel and copper are removed by chlorine leaching and recycled to the process. Precious metals are treated in the precious metals refinery for separation and recovery.

In both the Inco and Falconbridge electric furnace smelting processes about 50% of the sulphur in the concentrate is removed in a continuous 10% SO_2 gas stream from the fluid bed roasters. Falconbridge recovers this SO_2 in an acid plant. At Inco Thompson, the SO_2 is vented to atmosphere because of acid marketing constraints related to the smelter's location. In both processes, electric furnace gases are too dilute and converter gases are too variable for efficient conversion to acid. These are therefore vented to atmosphere and amount to about 10% and 25% respectively of the original sulphur in the concentrate. About 12% of the original sulphur remains with the matte product, while the remaining relatively small percentage is fixed in the discard slag.

Inco has developed and is presently piloting a new roast/electric smelt process which is capable of removing 80% of the total sulphur in the concentrate in a 10% SO_2 gas stream from the fluid bed roasters. The electric furnace is operated under reducing conditions to produce a high grade (about 50%) Ni/Cu matte and a discard slag. Conventional batch converting is then used to produce a final matte grade of 75% to 80%. Converter slag is recycled to the electric furnace. There may still be problems on a commercial scale related to metal losses in the slag at the matte grades required. The process however has the potential to efficiently recover 80% of the total sulphur as acid, while limiting SO_2 emissions to about 10% of the sulphur. The latter is divided equally between electric furnace gases and converter gases which are too dilute and too variable respectively for efficient conversion to acid.

In general the electric furnace smelting processes facilitate the recovery of 50 to 80% of the sulphur in the concentrate in a high strength SO_2 gas suitable for conversion to acid. However, the processes are energy intensive, in that they do not make efficient use of the sulphur fuel in the concentrate to minimize total energy requirements. Nevertheless, the new Inco process has the potential to limit SO_2 emissions to 10% or less of the total sulphur in the concentrate while facilitating the recovery of 80% of the sulphur as acid from the roaster off-gases. In addition, most of the sulphur in the final Ni/Cu matte (about 10% of total S) can also be efficiently recovered in the refining process, either as elemental sulphur if the matte electrorefining or the matte leach process is used, or as acid if the matte separation process followed by fluid bed roasting to nickel oxide is used. Since most of the energy used in the smelting process is electrical, with very little requirement for high cost carbonaceous fuels, the new Inco process may be a reasonable compromise between environmental and energy considerations.

Flash smelters

Outokumpu flash smelters are used for processing nickel concentrates in Norway, Australia and Botswana, and a new plant is being built in the USSR (29,38). In the process, predried concentrate and flux are flash smelted with preheated air or oxygen enriched air to produce a 45% to 60% Ni/Cu matte, and a 10% or greater SO_2 gas which is readily amenable to recovery in an acid plant. The matte is further oxidized in conventional converters to produce a final matte grade of 75% to 85% which is suitable for refining. Smelter and converter slag are cleaned in electric furnaces with the addition of coke and sulphidizing agent (usually pyrite) to recover Ni/Cu values prior to discard.

A sulphuric acid leach process followed by purification and electrowinning is used to recover refined nickel, copper and cobalt from the matte. Precious metals are recovered from the leach residues.

Using oxygen enriched air, the Outokumpu flash smelter removes up to 80% of the sulphur in the concentrate as a continuous high strength SO_2 gas readily amenable for recovery in an acid plant. The process makes reasonably efficient use of the sulphur fuel in the concentrate, and at high oxygen enrichments, limits external energy requirements primarily to that required for electric furnace slag cleaning and for pre-drying the concentrate.

Inco's flash smelter has also been tested on an experimental basis on nickel concentrates (39). As with copper, the Inco flash smelter using 100% oxygen can operate autogenously and produce 25% to 45% Ni/Cu matte and discard slag. Up to 60% of the sulphur in the concentrate can be removed in a continuous high concentration SO_2 gas (greater than 70%), which can be sold as liquid SO_2 or readily converted to acid. The high SO_2 concentration and high oxygen utilization can also facilitate conversion to elemental sulphur if this option should prove to be desirable due to sulphuric acid marketing problems (40).

In both the Inco and Outokumpu processes the flash smelter matte is further processed in batch converters to remove iron and sulphur and produce the required matte grade for refining. In the Inco process the converter slag can be recycled to the flash smelter, thus no electric furnace is required for slag cleaning. However, because of the lower matte grade, about 30% of the total sulphur in the concentrate will be removed during batch conversion compared with about 10% in the Outokumpu process. The Inco process therefore, although more energy efficient, is not as desirable for SO_2 emission control. There is also some doubt that high recoveries of nickel and copper can be maintained without slag cleaning, particularly at the matte grades required to ensure 60% sulphur removal in the smelter. Again, what is required is a continuous conversion process to facilitate SO_2 capture from the converting step, and thus allow greater flexibility in the smelter operation.

Continuous smelters/converters

There are no continuous smelting/converting processes for nickel presently operating on a commercial scale. A number of processes have been proposed in the past and at least one, the WORCRA process, has been investigated on a pilot scale (41). Over the past three years Inco has been investigating, under a government shared cost program, a continuous smelting/converting/slag cleaning process for nickel on a pilot scale at Port Colborne. The investigation is continuing.

Hydrometallurgical Processes for Nickel

Sherritt Gordon ammonia leach process

Although many hydrometallurgical processes have been investigated for extracting nickel from sulphide concentrates only the Sherritt Gordon process is presently in commercial production, with plants operating in Canada since 1954 and in Australia since 1971 (29,42,43). In the Sherritt process nickel sulphide concentrate is oxygen (air) pressure leached in an ammoniacal solution at 0.8 MPa (115 psig) and 93°C (200°F) in a continuous two-stage process to dissolve nickel, copper and cobalt, convert iron to an insoluble hydrated oxide and convert sulphide sulphur to ammonium sulphate. Byproduct copper is recovered as copper sulphide from the leach solution by boiling to remove ammonia, followed by H_2S precipitation. The nickel bearing solution is then further oxidized at much higher temperatures and pressures to convert thiosalts to sulphates. Nickel is recovered as a metallic powder by pressure hydrogen reduction in batch autoclaves. Under the conditions used cobalt remains in solution and is recovered later, along with residual nickel by H_2S precipitation. The metal free solution is then evaporated to crystallize ammonium sulphate which is sold as fertilizer. The economics of the process are greatly affected by the marketability of this fertilizer byproduct.

The Sherritt process is ideal for producing controlled particle size nickel powders for powder metallurgy applications which can command a premium price. It can also operate economically at much lower throughputs (15 000 to 30 000 t/a Ni) than conventional nickel smelters. The process also recovers copper and cobalt byproducts, although cobalt recovery is rather poor. Precious metals remain in the leach residues and are difficult to recover. For this reason it is preferable that the concentrates used in the process be relatively free of precious and platinum group metals. A modified version of the Sherritt process in commercial operation in South Africa, does recover platinum group metals from the leach residues (44). However the feed to the process is a high grade (low iron) Ni/Cu matte which facilitates recovery due to the very high concentration of the precious metals in the leach residues.

Although the Sherritt process has no environmental constraints related to SO_2 emissions, it does require a high cost reagent (ammonia) and depends on adequate markets for the ammonium sulphate fertilizer byproduct. Ammonium sulphate is not an ideal high demand, high value fertilizer. This, coupled with the difficulty in recovering precious and platinum group metals, makes it unlikely that the process can be used extensively in Canada to resolve the SO_2 problem, particularly in inland locations where transportation costs seriously affect the fertilizer byproduct marketability.

Sumitomo oxygen pressure leach process

The Sumitomo process began commercial production in 1978 on relatively small scale (2500 t/a Ni) to recover nickel, cobalt and copper from a 25% Ni matte produced from a laterite ore by a sulphidization technique (45). Although relatively low in nickel and copper, compared with high grade converter mattes, it contains nearly 12% cobalt and less than 2% iron. The feed to the process therefore contains nearly four times as much Ni/Co/Cu values as normal nickel concentrates from sulphide ores and the iron content is less than 10% of that normally associated with conventional nickel concentrates. Therefore, the process is not directly applicable to conventional high iron nickel sulphide concentrates.

The process consists of oxygen (air) pressure leaching which converts the sulphides in the matte to metal sulphates. This is followed by purification to remove iron and other impurities and recover copper as sulphide by H_2S precipitation. The nickel and cobalt sulphates in solution are converted to chlorides and separated by solvent extraction techniques. High purity nickel and cobalt are then recovered by electrowinning in chloride electrolysis cells. Chlorine from electrolysis is recycled to the process. Sulphates are initially converted to ammonium sulphate during the first solvent extraction step, and are eventually disposed of as gypsum produced by lime treatment of the ammonium salt to recovery and recycle ammonia.

Freeport nickel pressure leach process

A second commercial hydrometallurgical plant for nickel using a very similar oxygen pressure leach process, was built and operated in the USA in 1960. The feed to the plant was also a very low iron Ni/Co matte (55% Ni) produced from a Cuban laterite ore. Sulphides in the matte were converted to metal sulphates and after purification nickel was recovered by hydrogen pressure reduction from ammonium sulphate solution, similar to the Sherritt process. Cobalt and ammonium sulphate byproducts were also recovered. The plant is no longer in operation.

Both the Sumitomo and Freeport processes are really refining processes for nickel mattes (rather than extraction processes for nickel concentrates), somewhat similar to the Falconbridge chloride leach processes for high grade mattes described earlier (34,35,36,37). The processes have not been proven on conventional high iron nickel sulphide concentrates, and are unlikely to be viable because of the problems associated with iron removal and precious metal recovery. The acid sulphate processes (Sumitomo and Freeport) are particularly undesirable because sulphides are converted to sulphates rather than elemental sulphur in the extraction step, and no saleable byproduct is produced.

Inco process for pyrrhotite

Inco's plant for producing nickel, iron ore and sulphuric acid from pyrrhotite was built in 1955 (28,29). Pyrrhotite contains less than 1% nickel, so that nickel production is only a minor part, (less than 1% by weight) of the total plant output. Fluid bed roasters are used to convert most of the sulphides to SO_2 which is recovered in an acid plant. The roaster calcine, which is primarily hematite with nickel present as both ferrite and sulphide is selectively reduced with natural gas in rotary kilns to convert the nickel to a leachable form and the hematite to magnetite. The product is then countercurrently leached in ammonia, with CO_2 addition from the reduction kiln off-gases. Magnetic separators are used for solid/liquid separation and the leached magnetite is pelletized and sintered on a travelling grate to produce 67% Fe pellets.

The nickel bearing solution is treated to recover copper and cobalt, and ammonia is boiled off for recycle to the process. Nickel is precipitated as carbonate and decomposed to produce a reactive nickel oxide. Nickel recovery in the process is about 80%.

The main problem with the process has been the inability to obtain adequate markets and netbacks for the acid and iron ore products. Nickel recovery is not very good, and has led to some marketing problems for the iron ore due to the residual nickel content.

Other processes for pyrrhotite

In 1972, Falconbridge built a processing plant to recover ferronickel and sulphur from pyrrhotite containing 1.5% nickel (46,47). The pyrrhotite is roasted, pelletized and reduced with coal/coke and natural gas in a very large kiln to produce a ferronickel product which is at least 95% metallized. The roaster gases are piped to an adjoining Allied Chemical Plant where SO_2 is reduced to elemental sulphur using natural gas as the reductant.

The reduction kiln never achieved design capacity and the plant was shut down after less than one year's operation, partly because of the kiln problem, but also because reduction of SO_2 to sulphur with natural gas had become uneconomic.

A sulphation roast process has also been used for pyrrhotite by Falconbridge (29) and is presently still being used by Outokumpu in Finland (48). Controlled temperature roasting is used to convert iron sulphides to hematite and SO_2, while nickel and other non-ferrous metals are converted to soluble sulphates which are readily leached from the calcine. Nickel, copper and cobalt are recovered from the sulphate leach liquor by H_2S precipitation. SO_2 is converted to acid and the iron is sold for steel production.

Similar to other roasting processes the economics depend mainly on adequate nearby markets for acid and iron ore.

Numerous hydrometallurgical processes have been proposed for treating pyrrhotite. A comprehensive review of the possible alternatives was published in 1975 (49). Although most of the processes are technically feasible the most economically attractive for Canada are those that efficiently convert sulphide to elemental sulphur while minimizing problems associated with the removal of iron from solution and conversion to hematite. The process should also be capable of obtaining high recoveries of the nickel and associated byproducts. The most promising are those that are carried out in chloride media, to facilitate high conversion of sulphides to sulphur at atmospheric pressure. The iron chlorides can then be selectively oxidized to hematite while nickel and other non-ferrous metals remain as chlorides and can be recovered by leaching. The sulphide and chloride oxidation steps can be carried out either hydrometallurgically (50,51) (aqueous media) or pyrometallurgically (52) (non-aqueous media). The latter is particularly attractive because the process can be autogeneous and iron and sulphides are oxidized at moderately high temperatures in a dry system, thus minimizing solids/liquid separation problems for iron and sulphur recovery.

Zinc Smelting and Refining

Canada has four zinc smelter/refineries at a capacity of about 665 000 t/a. All four use the roast-leach-electrowin (RLE) process and three have acid plants for SO_2 emission control. One producer (HBMS) does not have an acid plant, primarily because SO_2 gases are too dilute, but also because of H_2SO_4 marketing problems resulting from their isolated location.

Noranda and Kidd Creek have jarosite circuits for iron removal, while Cominco and HBMS recycle zinc plant leach residues to the lead and copper smelters respectively to recover residual zinc, plus lead, copper and precious metals originally present in the zinc concentrates.

Cominco also has the first commercial hydrometallurgical zinc plant in the world, at a capacity of 30 000 t/a (53). A second plant, using the same technology is being built in Ontario for Kidd Creek (54).

Conventional Zinc Concentrate Processes

Roast-leach electrowin (RLE)

This process is used extensively throughout the world (55,56,57,58). It uses autogeneous fluid bed roasting at high temperatures to convert all of the sulphur in the concentrate to SO_2, which is removed as a continuous high strength (10%) gas readily amenable to recovery in an acid plant. The roaster calcine is leached in sulphuric acid (spent electrolyte), usually in two stages to ensure recovery of zinc from ferrite which is formed during the roasting step. The second stage leach is carried out at high temperature and high acid concentration to decompose the ferrite. Iron and minor impurities are removed by jarosite precipitation and the leach liquor is recycled to the first stage dilute acid leach (59,60,61). First stage leach solution is pH adjusted for final iron removal and further purified by cementation techniques to recover cadmium and copper. High purity cathode zinc is produced by electrolysis of zinc sulphate solution. In some cases leach residues are processed in lead or copper smelters to recover lead, copper and silver.

Sherritt Gordon Pressure Sulphuric Acid (PSA)

This process utilizes oxygen pressure leaching in sulphuric acid (spent electrolyte) at 150°C to solubilize zinc and convert metal sulphides to insoluble elemental sulphur (62,63). Pyrite is not significantly attacked in the process. The leach liquors are pH adjusted to remove iron, arsenic and other impurities, and are further purified by conventional cementation techniques to recover cadmium and copper. High purity cathode zinc is produced by electrolysis of zinc sulphate solution. Spent electrolyte is treated to control chloride levels prior to recycling to leaching. Elemental sulphur can be recovered from the leach residues. In some cases the residues can be further processed in lead or copper smelters to recover lead, copper and silver.

For high grade (more than 50%) zinc concentrates the RLE and PSA processes have no serious problems. Zinc recoveries are at least 95%, and the choice of processes provides industry with the flexibility to produce either acid or elemental sulphur as a saleable byproduct.

The PSA process can also handle low grade zinc concentrates. However, neither process can efficiently recover high value lead, copper and silver byproducts from the leach residues. Silver is particularly troublesome, because many zinc concentrates contain significant silver values which are difficult to recover economically from the high iron leach residues.

Bulk Concentrate Processes

Imperial smelting process (ISP)

This is the only commercial process which can treat bulk concentrates containing primarily zinc and lead, with minor amounts of copper and precious metals (64). There are thirteen ISP plants throughout the world. One was built in Canada, but did not prove to be viable on bulk concentrates and was eventually converted to a lead smelter. The plants operate by blending a specific ratio of relatively high grade zinc and lead concentrates to provide the major portion of the feed to the process. Significant amounts of "true" bulk concentrate can then be blended and recovered.

The blended concentrates are mixed with recycle sinter and roasted with air in an up-draft sintering machine to convert most of the sulphides to SO_2 and produce a zinc/lead/iron oxide sinter, which is crushed, screened and fed to a blast furnace with sufficient coke and flux to produce a molten lead bullion and molten discard slag which are both tapped off at the bottom of the furnace. Zinc leaves the furnace as a vapour and is condensed in a splash condenser by dissolving in a recycle stream of hot molten lead. The molten lead is cooled in launders causing zinc to crystallize out of solution and float on the surface of the molten lead. The zinc is skimmed off, while the cooled lead is recycled to the condenser. Most of the zinc produced is sold as "prime western" grade containing up to 1.5% lead. High grade zinc can however be produced by a distillation process in reflux columns. Zinc and cadmium are distilled and recovered separately, while molten lead is tapped from the column discharge. Most ISP plants have a zinc refining facility to meet the demand for high zinc.

SO_2 from the sintering furnace is recovered in an acid plant and the lead bullion from the blast furnace is usually electrolytically refined to produce a high purity product and recover precious metals from the anode slimes. Copper in the lead bullion is recovered as a lead/copper dross.

Today, ISP plants obtain reasonable recoveries (about 95%) of zinc, lead and precious metals, and, by utilizing waste heat sources to preheat blast furnace air and coke, are reasonably energy efficient. However, the high temperature operation, the sinter size control required for the blast furnace, the large recycle of sinter and lead required for temperature control in the sintering machine and zinc condenser respectively, and the resultant materials handling and fugitive emission control problems do not make the process attractive for meeting present and future environmental and health and safety regulations, particularly for lead and possibly for arsenic. In addition, copper and cadmium recoveries in the process are generally poor, and high purity zinc cannot be produced without the additional high temperature refining step. The process also does not have the flexibility to treat a wide range of bulk concentrates, with different grades and zinc/lead ratios. High levels of undesirable impurities can also be a problem, for both emission control and zinc purity.

Emerging technologies for bulk concentrates

The N.B. Research and Productivity Council's (RPC) sulphation roast leach (SRL) process (65) and Sherritt Gordon's modified pressure sulphuric acid (MPSA) process (66) are two very promising Canadian technologies. Both produce electrolytic zinc and high grade lead/silver and copper byproducts suitable for further processing in lead and copper smelters. Both can operate on a wide range of bulk concentrate grades, metal ratios and impurity levels and have no significant emission control problems.

In the SRL process, bulk concentrate is roasted at a controlled temperature to convert iron sulphides to hematite and SO_2, and non-ferrous sulphides to sulphates. The SO_2 is recovered in an acid plant and the roaster calcine is leached in spent electrolyte in a two-stage process to dissolve all the zinc and copper. Neutral leach liquor from the first stage leach is purified by conventional methods, copper and cadmium are recovered, and cathode zinc is produced by conventional electrowinning. However, due to sulphate build-up the spent electrolyte cannot be completely recycled and a lime neutralization circuit is necessary to maintain the sulphate balance and recycle zinc. The high acid leach liquor from the second stage leach, which contains high concentrations of iron and zinc from ferrite dissolution, is recycled to the roaster for temperature control and to decompose the iron sulphate to hematite and SO_2.

The lead is converted to lead sulphate and remains in the leach residues along with the precious metals. Lead and some of the silver are recovered by a brine leach, lime precipitation technique to produce a high grade (chloride free) calcium plumbate which is ideal feed to a lead smelter. The remaining silver is recovered either by flotation or by cyanidation, which will also recover gold if present.

In the MPSA process the bulk concentrate is oxygen pressure leached in two stages to dissolve zinc and copper and convert sulphides to insoluble elemental sulphur. Pyrite is not significantly attacked. The leach liquor is purified by conventional methods, copper and cadmium are recovered and cathode zinc is produced by conventional electrowinning.

Most of the lead is converted to lead jarosite and remains in the leach residues along with the precious metals, pyrite and elemental sulphur. The pyrite and elemental sulphur are removed by flotation and the lead jarosite is upgraded to lead sulphate by aqueous treatment with SO_2 at about boiling temperatures. The lead sulphate and retained precious metals are fed to a lead smelter for conversion and recovery.

Both processes can achieve very high (greater than 96%) zinc and lead recoveries from bulk concentrates. The SRL process can also achieve very high (97%) recovery of copper, regardless of the copper content of the bulk concentrate, and electrolytic grade copper can be recovered if economics so warrant. Copper recovery in the MPSA can be as high as 90% but requires an interstage precipitation if copper levels in the bulk concentrate exceed 1%. Precious metals recovery requires improvement for both processes, but particularly for the MPSA where losses up to 25% can occur during the flotation separation of the sulphides from the leach residues. More efficient and economical methods are still required for recovering the lead and precious metals from both the SRL and MPSA process leach residues. The recovery of sulphur from the MPSA process residues should also be investigated to improve the economics.

A number of aqueous chloride leach processes have been proposed for bulk concentrates (25,67,68,69). The processes use chlorine, ferric or cupric chloride at atmospheric pressure to solubilize non-ferrous metals and convert sulphides to elemental sulphur. Pyrite is not significantly attacked in the process. Iron and arsenic can be rejected by oxy-hydrolysis, and lead can be recovered by crystallization of lead chloride, followed by fused salt electrolysis. Zinc and copper can both be recovered by aqueous electrolysis from chloride media, or from sulphate media if the separation/purification step used is amenable to conversion of chloride to sulphate. Silver is also dissolved in the chloride leach and can be recovered by cementation. However, most of the gold remains in the leach residues.

A pyrometallurgical chlorination/oxidation process has also been proposed for bulk concentrates (70). Dry concentrate is "flash smelted" with chlorine in a continuous chlorination reactor to produce molten metal chlorides and elemental sulphur which is removed as a gas and condensed. The metal chlorides flow to a deferrination reactor where more chlorine is introduced to remove iron and gaseous ferric chloride. The remaining non-ferrous metal chlorides are granulated in air or oxygen to oxidize any remaining iron to hematite, prior to dissolution in slightly acid chloride media. Lead chloride is selectively crystallized for recovery by fused salt electrolysis. Zinc and copper chlorides are separated and purified by cementation or solvent extraction techniques. Zinc is recovered by aqueous electrowinning in chloride media, while copper is recovered either by cementation or electrolysis. Precious metals are also recovered by cementation. The ferric chloride is reacted with oxygen in a dechlorination reactor to produce a high purity hematite and chlorine for recycle to the process. Chlorine produced in the electrolysis cells also recycles to the process.

The process fully utilizes the sulphur energy in the concentrate and can convert essentially all of the sulphides in the concentrate to elemental sulphur, the non-ferrous metals to chlorides and the iron to hematite without requiring any external source of energy. The process also has the potential to recover all of the valuable components in the concentrate, including sulphur and iron as high value saleable products. However, much more laboratory and pilot scale research and development is required to determine the impurities, optimize reactor design and conditions for the pyrometallurgical steps in the process, and determine the most efficient solution purification and metal and sulphur recovery techniques.

Lead Smelting and Refining (Pyrometallurgical Processes)

Canada has two lead smelter/refineries at a total capacity of about 245 000 t/a. Both smelters use the sinter/blast furnace process and both recover SO_2 from the sintering furnace in an acid plant. Cominco uses electrolytic refining to produce a high purity lead product and recover valuable byproducts. BMS utilizes thermal refining techniques, and also recovers valuable byproducts. Cominco also has a slag fuming plant to recover zinc from the lead furnace slag.

Sinter/blast furnace process

Nearly all of the world's present primary lead production is produced using the sinter blast furnace process (71). The concentrates, mixed with recycle sinter, are roasted in a travelling grate sintering furnace to produce a lead/iron oxide sinter, which is crushed, screened and fed to a blast furnace. Sufficient coke and flux are added to produce a molten lead bullion and molten discard slag. SO_2 from the sintering furnace is recovered in an acid plant. The lead bullion is refined either electrolytically or thermally to produce high purity lead and precious metal and other byproducts.

The process suffers mainly from the high temperature materials handling and fugitive emission problems which make it very difficult to meet present and future in-plant emission regulations. The process also is not very energy efficient, since it does not make good use of the sulphur fuel in the concentrate.

Emerging pyrometallurgical processes for lead

A number of direct smelting processes (72) have been proposed for lead concentrates over the past few years and at least three processes (73,74,75) have been developed on a demonstration scale. All employ the concept of oxygen smelting, followed by slag cleaning to produce a lead bullion and discard slag. Therefore, the processes make good use of the sulphur fuel in the concentrate and are designed to minimize materials handling, off-gas volumes and fugitive emissions. KIVCET (73) and Outokumpu (74) use flash furnaces for smelting and electric furnaces for slag cleaning. Both use pure oxygen and produce very high SO_2 concentrations in the smelter gases. In the Outokumpu process the flash smelter and electric furnace are separated, while in the KIVCET process they are combined into a single unit. Both processes use coal or coke for reduction in the electric furnace.

In the QSL process smelting and slag cleaning are carried out in a single "Noranda type" reactor (75). The concentrate is first pelletized for feeding to the reactor, pure oxygen is injected through tuyeres in the smelting zone, and pulverized coal is injected in the slag cleaning zone. Combined off-gases from the reactor contain 15% SO_2.

All are capable of obtaining high recoveries of lead (98%) and associated byproducts. The KIVCET also claims to recover zinc. Process gases are readily amenable to SO_2 recovery in an acid plant, and the SO_2 concentration in the Outokumpu and KIVCET processes is high enough to produce liquid SO_2.

The ability of the processes to handle lead bearing residues and low grade or high impurity concentrates has not yet been adequately demonstrated. A better understanding of the factors controlling impurity and byproduct distribution in the molten slag-metal systems is required. The downstream effect on refining processes, particularly electrolytic, may also pose problems which will require investigation of the refining technologies to minimize impurity related problems and maximize byproduct recovery while meeting in-plant emission regulations.

Hydrometallurgical Processes for Lead

One of the most attractive hydrometallurgical processes for lead is the ferric chloride leach process (76,77,78). Although there are a number of variations to the process it consists essentially of selective ferric chloride leaching to solubilize lead as chloride and convert sulphides to elemental sulphur. The lead chloride is then selectively crystallized and lead is recovered by aqueous or fused salt electrolysis in an electrolytic cell. Chlorine from the electrolysis is recycled to regenerate ferric chloride.

The process has been demonstrated on a small scale but considerably more R & D is required to develop a commercial aqueous or fused salt cell, and to develop viable methods for controlling impurities and recovering byproducts (79,80).

PROBLEMS

HEALTH AND SAFETY

Workplace health and safety has become a major industrial concern, particularly as knowledge of occupational diseases increases (3). The direct cost of accidents and industrial diseases is increasing as compensation is raised to reflect broader definition of compensable diseases and wage increases.

Accident rates vary according to type of mine and occupation within mines (3). Open pit mines are safer than underground, with the fatality rate of the former only about one third that of the latter. The fatality rate in underground silver mines is nine times that in underground nickel mines. Mills and smelters provide a safer working environment, followed by shops and surface facilities. Accident rates vary among companies. Although size appears to be a factor - larger companies presumably can afford safer working environments and better safety programs - other factors may contribute.

Although efficient dust collecting systems are used in most crushing plants, these have not been applied to dust control in concentrators. The two main sources of dust in the atmosphere are at transfer and feed chutes of ore feed conveyors and from mineralized bubbles bursting and releasing dust and chemical vapours above the flotation cells. Handling of deadly and corrosive chemicals also presents dangers. Surface active agents used to suppress dust before the ore is beneficiated may affect the chemical properties of the minerals and cause grave problems. Workers in smelters and refineries are also exposed to dust, fumes, high temperatures, corrosive liquids and labour intensive tasks which create major health and safety problems, a great number of which still need to be resolved.

ENVIRONMENT

Like most industrial activities, mineral processing can disrupt the environment (3). After the resource is mined, processing operations can affect water and air quality by dispersing contaminants in tailings or into the atmosphere. Acid rain, caused by the emission of sulphur dioxide and nitrogen oxides into the air, is one of the more serious environmental problems.

Although its ecological effects and health hazards are not yet fully understood, acid rain has undoubtedly contributed to the acidification of lakes and streams, is thought to be adversely affecting some agricultural crops and forest growth, and has the potential to contaminate drinking water systems. Unless the loading of acid rain into the environment can be arrested and reversed, this environmental degradation will continue.

Generally, sulphur dioxide is responsible for about two thirds of the acidity in precipitation in Canada (reverse in USA); the other one third is from nitrogen oxides. Sulphur dioxide is the prime target for control measures because it is the major source of acidity and also the more tractable of the two sources (3).

The problem is particularly acute in Eastern Canada and the northeastern USA. The major sources of sulphur dioxide in Eastern Canada are nonferrous metal smelters, which produce more than 40% of the region's total emissions. In the USA, the major sources are fuel-fired electrical generating stations, which account for well over 60% of all emissions.

It should be noted that sulphur dioxide emissions at some Eastern Canadian nonferrous operations have been significantly reduced over the past decade. Emissions of the Inco smelter in Copper Cliff were reduced from 5500 t/d in 1969 to 2270 t/d in 1980. This smelter now successfully contains about 70% of the sulphur in its feed. The Falconbridge nickel smelter, which emitted about 940 t/d in 1969, now emits about 420 t/d. Falconbridge's emission control system contains about 80% of the sulphur in the ore processed. Other smelters control well over 85% of the sulhpur in their ore - zinc smelters at Vallyfield, Quebec, and Timmins, Ontario, a lead smelter at Belledune, New Brunswick, and the copper smelter near Kamloops, British Columbia.

More than 50% of the sulphur dioxide that falls in Eastern Canada comes from the USA, while Canada's contribution to total American depositions is only about 10%. To all intents and purposes, Canada is a recipient of transboundary pollution - it is subjected, in absolute and relative terms, to more pollution than it inflicts upon others.

This is a bilateral problem that can be solved only through cooperative action with the USA. In 1980, a Memorandum of Intent was signed by the two governments setting up the framework for negotiation of a transboundary air pollution agreement, the "core" of which would be a mutual obligation to reduce pollutant emissions. This obligation is consistent with the responsibility of states under international law to ensure that activities within their jurisdiction or control do not cause damage to the environment of other states.

The direct costs of reducing sulphur dioxide emissions are expected to be high. The Department of the Environment (DOE) estimates that a capital investment of $620 million (1980$) would be required by Eastern Canadian nonferrous smelters to reduce emissions by 57%. The cost of an 80% reduction is estimated to be $1000 million. The net increase in annualized operating and capital costs for the 57% reduction is estimated at $120 to $150 million (1980$) - equivalent to $0.33 to $0.44/kg of nickel and $0.11 to $0.18/kg of copper. The annualized operating and capital costs for an 80% reduction are estimated to exceed $200 million.

DOE estimates that, operating at capacity, an 80% reduction in sulphur dioxide emissions from the Inco copper-nickel smelter at Sudbury would require an investment of $450 million (1980$), with a net increase in annual capital and operating costs of approximately $100 million. This is equivalent to about $0.44 to $0.66/kg of nickel.

In addition to sulphur dioxide emissions, mineral processing activities can affect the quality of the air, soil and water in a number of other ways. In terms of air quality, metallurgical plants can emit heavy metals, inorganic and organic compounds, and many gaseous emissions other than sulphur dioxide. Technology offers the promise of nearly pollution-free processes, but until these are fully developed and commercially proved, governments will, as in the case of sulphur dioxide emissions, face trade-offs between industrial activity and environmental quality.

The degradation of the terrestrial environment is also of concern. However, because of the remoteness of many mining and milling operations, the visual impact on the landscape - aesthetic pollution - is minimal.

The most significant aspect of soil pollution is potential groundwater contamination from tailings disposal and effluent impoundments containing cyanide, heavy metals and thiosalts, with possible seepage and migration to underlying aquifers, or surface runoff that could pollute major drainage basins. In the fragile northern environment, with slow natural recovery rates due to low temperatures for most of the year, this is an important consideration.

Tailings disposal in lakes can destroy fish habitat, contribute to eutrophication and generally upset the local ecology. In addition, oceans are susceptible to pollution from tailings disposal or other runoffs arising from insufficient water circulation and movement in coastal areas.

COMPETITIVENESS

The competitive position of the Canadian mineral industry on international markets will, to a large measure, determine its future performance and its contribution to the Canadian economy. Canada's competitive position is based on relative costs. The lower the cost associated with a mineral deposit in Canada, the more price-competitive it is relative to other deposits in the world (3).

The most important factors affecting competitive position are ore grades and types. Copper illustrates the importance and trend in ore grades and types.

The average ore grades for all known Canadian reserves of copper fell significantly from 1960 to 1976, compared with ore grades in other countries. This is not an altogether unwelcome development, since it implies that technological advances in the 1960s and 1970s greatly reduced the economic cut-off grade for large-scale, open-pit porphyry copper deposits which, in turn, had the effect of significantly expanding Canada's copper reserves. The fact remains, however, that ore grades in Canada are considerably lower than the world average. The average ore grade for producing mines in Canada is 0.7%, among the lowest in the world and significantly below the world average of 1.03%. In other words, the grade of copper ore currently being mined in Canada is, on average, below that of all major competitors except the Philippines, Papua, New Guinea and Mexico. However, it should be noted that despite the high ore grades in Zaire and Zambia, mining and processing costs in those countries are high due to difficult geological structures and the metallurgical characteristics of the ore deposits (3).

The disadvantage of mining lower average copper ore grades can be offset by the valuable coproducts and byproducts contained in many Canadian deposits.

There are essentially two kinds of copper orebodies in Canada - porphyry deposits found in British Columbia, which are mined by open pit methods and sulphide deposits found mostly in

Manitoba, Ontario and Quebec, which are mined by underground methods. The sulphide deposits contain significant quantities of gold, nickel, cobalt and zinc, and to a lesser extent, silver, the platinum group metals and lead. For some mining operations, the total value of coproducts and byproducts exceeds the value of the copper. However, this is not true for the porphyry copper deposits in British Columbia. These contain few coproducts and byproducts, although some deposits do contain signficant quantities of molybdenum and gold and, to a lesser extent, silver. Because of this lower coproduction of metals, the porphyry deposits are more vulnerable to competitive pressures than the massive sulphide deposits.

During the 1980s and 1990s, the average ore grade at Canadian mines is expected to fall more than those of our major competitors, thus increasing Canada's disadvantage on the basis of reserve availability and ore grade.

Remoteness of mine location will continue to grow in importance as a factor determining competitiveness in the mineral industry. This is particulary true in Canada, where many major new mineral developments are likely to occur far from existing infrastructure and be subject to severe climatic conditions. In addition to higher infrastructure costs, unfavourable geographic location can mean longer development and preproduction phases because of the long distances from supply sources. It is sufficient to emphasize here that infrastructure for a major mineral development in Canada's North can represent well over half the total capital costs of the project.

In contrast, some relatively unexplored or more richly endowed third world countries might be able to develop deposits in future that will be more favourably located. This could happen in Chile, Peru and the Philippines. It is therefore likely that Canada will experience a comparative disadvantage in the years to come, as more and more of its developments take place in remote areas.

Labour is the single most important component of operating costs for Canadian mineral producers. While wage developments in the mineral sector have generally reflected those in the rest of the economy, they have consistently exceeded the average for the total industrial sector, with the gap widening in recent years. This gap is also widening between Canada and its major third world competitors.

Since the dramatic increase in world oil prices, energy costs have assumed great importance for all Canadian industries, including the mineral sector. Energy, as a component of cost, has made and will continue to make a positive contribution to the competitive position of the Canadian mineral industry during the 1980s. There may be periods when oil prices rise faster in Canada than elsewhere, but over the decade, oil prices will not exceed world prices, given that the Canadian price for "old" oil will not exceed 75% of international prices. Moreover, the parity price of natural gas should decline. Furthermore, Canada is well endowed with other sources of energy, such as hydro power and coal, and has an excellent nuclear energy potential. The competitive edge on energy costs, however, will vary among mineral commodities - the more energy-intensive the commodity, the greater will be its competitive edge.

Canada seems to have an edge in terms of exploration, mining and processing technology compared with many other mineral producers. But to remain competitive, Canada will have to stay in the forefront of technology development, and new technology must be continuously adopted by industry.

The major factors affecting the competitive position of the Canadian mineral industry, relative to competitors, are summarized below, together with their impact on the industry during the 1970s and their expected impact through the 1980s (3):

SELF SUFFICIENCY

Several recent events such as the formation of OPEC, political tensions in Africa and Southeast Asia, the rapid reversal of the USSR from exporter to importer of some important raw materials, have heightened Western anxieties regarding the security of mineral and metal supplies.

Factors	Impact on competitiveness	
	1970s	1980s
Ore grade	Slightly negative	Slightly negative
Type of ore, e.g., co-products and byproducts	Strongly positive	Strongly positive
Location	Slightly negative	Moderately negative
Capital costs	Neutral	Slightly negative
Labour costs	Strongly negative but with improving trend	Strongly negative
Energy costs	Strongly positive since 1973	Moderately positive
Exchange rate	Strongly positive since 1976	Strongly positive in the absence of appreciation
Taxation	Slightly negative	Slightly negative
Technology	Strongly positive	Strongly positive
Environment	Neutral	Slightly negative
Tariffs and trade barriers	Strongly negative for processed minerals	Strongly negative for processed minerals

Tables 4 and 5 outline the materials for which Canada has no apparent import reliance and for which the import reliance is due to their processing abroad (3).

Table 4 - Materials for which Canada has no apparent import reliance

Lightweight aggregates: Clays, shale and slag
Asbestos
Barite
Bismuth
Cadmium
Calcium
Cement
Clays: common
Cobalt
Copper
Gallium
Gold
Gypsum and anhydrite
Indium
Iron ore
Iron and steel
Lead
Lime
Magnesium
Nepheline syenite
Nickel
Potash
Pyrophyllite
Salt
Selenium
Silicon: metal
Silver
Sodium sulphate
Sulphur
Tellurium
Titanium dioxide (ilmenite)
Zinc

Sources: Resource Strategy and Economic Analysis Branch, EMR, based on numerous sources.

Table 5 - Import reliance entirely due to the processing of domestic minerals abroad (content of exported ore and concentrates greater than content of imported material)

Antimony
Columbium
Molybdenum
Platinum group
Silica: artificial abrasives
Tantalum
Tungsten

Sources: Resource Strategy and Economic Analysis Branch, EMR, based on numerous sources.

As far the base metals and precious metals are concerned, the major problem is in the further processing of some of our mineral resources.

Further processing of mineral resources has been a continuing objective of Canadian public policy, because it provides employment, increases value added, promotes regional development and increases tax revenues (3). Most provincial mines acts require processing of resources before export but contain grounds for waiving the requirement. In 1974, the federal government amended the Export and Import Permits Act to support upgrading of resources. In addition, the extent of further processing in Canada is a criterion used by FIRA in assessing mineral development proposals.

Despite these measures and the policy that underlies them, about one half of Canada's nonferrous mine production still leaves the country in relatively unprocessed form. Although Canada's position at the Tokyo round of General Agreement on Tariffs and Trade (GATT) negotiations was based on the same policy, efforts to reduce foreign tariff barriers to more processed products have not been entirely successful (3).

On the surface, it would appear that Canada's best opportunity for further processing relates to the high volume of nonferrous mineral concentrate that is exported. If all this concentrate were smelted and refined in Canada, approximately 3300 jobs would be created in five new world-scale plants costing approximately $1800 million (1980$). The location of some of this activity in areas of high unemployment, such as northeastern New Brunswick and the Yukon, would also further the federal government's regional employment objectives.

Aside from direct employment benefits, additional processing capacity would offer marketing and cost advantages to industry. The products of a West Coast copper smelter and refinery could be marketed throughout the world, providing greater options for those British Columbia mines that are locked in to long-term contracts with specific Japanese smelters. A domestic smelter could treat mineral concentrates at lower concentrations of metal than foreign smelters will buy; savings in milling costs and additional revenues from lower cut-off grades would be possible with cooperation between mining and smelting operations. Moreover, Canadian smelters would obtain additional revenue from byproducts for which overseas smelters may not provide full credit when buying concentrate.

World smelting and refining above capacity is expected to last until the mid-1980s, after which substantial new and replacement capacity for primary non-ferrous metal processing probably will be required. Whether a significant portion of this new capacity will be built here depends on the extent to which Canada can remove or reduce the constraints that have prevented further processing in the past.

The 100 000-t Brunswick zinc smelter will come into production in the mid-1980s, to coincide with the expected upturn in market conditions for zinc metal. This expected improvement, combined with a favourable tax regime, should be sufficient to ensure that the smelter will be financially viable.

HIGH TECHNOLOGY

Concentrates are usually produced to specifications which have been determined to meet smelter needs or uses which meet general needs and most beneficiation techniques have been geared to satisfy these standards. In today's technology, pure materials are commonly required which are produced chemically. Most of these are industrial minerals although on the metallic side, iron ore superconcentrates have been used.

Given the high prices available for these higher purity chemically produced materials as well as advances in beneficiation technology, it would be advantageous to the industry to produce superconcentrates from ores which could be amenable to further treatment. Products which could be looked at are pure magnesia, high purity silica for fiber optics, etc.

Automatic process control of the unit operation in mineral processing and of the plant as a whole is still in initial development and growth, thus it is less advanced than other areas of automatic control engineering and science. The basis of this apparent technological lag lies in the peculiar difficulties which mineral treatment processes present to the process control engineer compared with the "cleaner" processes, e.g., in aerospace systems and even in most chemical process plants.

The relatively slow development of instrumentation is caused by the need to measure variables and the harsh environment. Accurate and maintenance free operation at reasonable cost for such simple but essential measurements as rock mass, flow and density of a slurry in a pipe are relatively recent achievements. Other measurement difficulties are on-stream analysis and on-stream measurement of size distribution of slurry streams. The necessary measurement principles were well established in the laboratory but took many years of development effort to translate into plant instruments.

Other measurements which are necessary for modelling and control are not now available. For example, in froth flotation, the concentration of specific ions and certain flotation reagents, reagent adsorption, flotation rate constants, and froth drainage rates are important state variables. While prototype instruments are being conceived, their practical application lies many years in the future. No well developed body of theory exists that would generate realizable optimal solutions to control problems in this difficult environment within a short time.

CONSERVATION OF RESOURCES

The increasing world demand for materials has resulted in resources being extracted, processed and consumed as products at ever increasing rates. In addition, waste products, generated from mining, processing and discard products have created serious environmental problems. It has become increasingly clear from both an environmental quality and resource conservation point of view, that this open-ended materials cycle is no longer acceptable, and must be replaced with a closed cycle incorporating recovery, reprocess and reuse of materials. The recycling of metals can also reduce considerably energy requirements, since the energy required to extract metals does not have to be expended again. This saving can range from 80% for iron to about 98% for magnesium.

Conservation policy planning must consider the three ways whereby more effective use of mineral and metal resources can be achieved: substitution, recycling and utilization of byproduct wastes. Canada has abundant resources of several important metals including copper, nickel, and lead. For these metals conservation practices probably should be directed towards developing improved recycling techniques. In other instances where our raw material resources are limited, attention should be given to finding suitable substitutes, thus conserving the resource in question for top priority applications. Substitution is of particular importance where the use pattern is such that recycling is impossible or difficult.

More specifically, there is not always the kind of cooperation required between mineral dressing plants and smelters which would permit a more optimal recovery of metal values from certain ores because smelters require certain grades which can only be obtained at lower recoveries.

Since mineral dressing plants and smelters can operate at a profit using existing processes and technology, there is a lack of incentive to optimize and develop new technology which would permit greater recoveries.

Fines and slimes have always created problems in processing. In many cases, it is easier to reject this fine fraction to tailings than to apply or develop processes to recuperate it.

Many byproducts are now rejected to tailings and slags because no technology exists to beneficiate them or the producer of a certain commodity has no expertise or is not interested in the byproducts present in the ore or concentrate.

Finally, valuable resources are lost to the country by shipping byproducts contained in concentrates to foreign buyers who offer very little or no payment for these byproducts while they recover them for their own use.

CONSERVATION OF ENERGY

The mineral industry is energy intensive, expecially at the smelting and refining stages. As a proportion of direct operating costs, energy makes up to 30 to 35% for zinc and 20 to 25% for copper.

Until quite recently, the popular belief was that Canada had abundant readily available energy resources. Recent studies conducted by the National Energy Board and EMR have done much to dispel this belief and suggest that Canada could be faced with an energy supply problem in the future. Thus it is imperative to encourage still greater conservation by implementing extensive programs to eliminate energy waste and to improve energy technology in the base metal industry.

Canada is fortunate with regard to electric power in that it has substantial reserves of uranium and coal for thermal electric generation. In addition, there are still a few undeveloped hydroelectric power sites. This situation should encourage industry to switch form oil to electricity whenever possible.

DIVERSIFICATION

Although Canada is the world's third largest mineral producer, domestic production of mining equipment and machinery is disproportionately small, providing less than half the mining industry's requirements and leaving the rest of this large market to foreign manufacturers (3). Canadian imports of mining machinery and equipment in 1980 totalled about $640 million, 53% of all such purchases by the mining industry.

The growing need for energy efficiency that will rapidly make the existing technology and equipment obsolete, stricter environmental regulations, increased automation and controls should offer wide opportunities for diversification.

LABOUR

Industry faces a recurring shortage of skilled people. If several large-scale projects are underway at the same time, e.g., petroleum, pipe-lining and mining, a greater demand for many of the same skills will be created (1). This will happen as the once-large youth group of the population ages, as early retirement becomes more the norm, and as competition becomes more effective in other countries for skilled trades people.

The mining industry is vulnerable in such periods of shortages, partly because of the remoteness of location and partly because of some misunderstanding of the industry in the public mind.

This developing trend to critical labour shortage could impair the industry's goals for growth and expansion, and hence have an adverse impact on the Canadian economy generally.

There is a singular difficulty of attracting and retaining employees in remote areas (particularly in Canada's northern territories). Traditionally, mining companies and governments in their employee programs have offered extra benefits such as subsidized housing and vacation travel allowances. Although the government is currently re-examining the tax status of these allowances, the industry feels strongly that some exceptions must be made to exempt them from tax, in order to make it easier for employers to attract and retain staff at northern and remote locations.

CURRENT RESEARCH AND DEVELOPMENT

Many performers around the world have been involved in the last few decades in R & D on hydrometallurgy and pyrometallurgy of base metals and to a much lesser extent on the mineral dressing of these ores. The present downturn in the mineral industry has cancelled a number of Canadian and other projects and shelved many more. Because of this, it is impossible to come up with accurate figures, but the following is a short list of some of the past, present and most likely future performers and some of the recent past and ongoing work.

PERFORMERS

CANMET, through its Mineral Sciences Laboratories, expended about 150 PY in the last few years, on a large project on complex, fine-grained sulphide ores.

Centre de recherches minérales in Quebec City, with a staff of 130 persons, has been involved in some confidential base metal R & D.

Noranda Mines Ltd., through its Research Centre in Pointe Claire, Quebec, (approximately 100 persons) and at its many mines, smelters and refineries, carry out many short and long range base metal projects.

Inco Metals Co., through its Research Centre in Sheridan Park in Mississauga, Ont., (approximately 150 persons) and its many mines, smelters and refineries, also are involved in many short and long term R & D projects.

Falconbridge Limited, through its research department (approximately 100 persons) support R & D activities.

Sherritt Gordon Mines Limited, through its research department (approximately 75 persons) is involved in extensive work.

Cominco Limited, through its Research Centre at Sheridan Park, Mississauga, Ont., (approximately 100 persons) and its mines, smelter and refinery, carry out extensive development work.

Ontario Research Foundation carries out contract work for the base metal industry.

B.C. Research Foundation is involved in bacterial leaching of sulphides.

New Brunswick Research Productivity Council is involved in fine-grained complex sulphide ores.

Lakefield Research, through its pilot plant facilities, services the base metal industry.

Dominion Engineering Works Ltd. is involved in the design and construction of large mills servicing the base metal industry.

Universities, mainly, Laval in Québec City, Alberta in Edmonton, British Columbia in Vancouver and to a lesser extent, Queen's in Kingston, and Toronto in Toronto, are involved in graduate student research and contract work for governmental or industrial concerns.

Most mines beside those listed above, expend a portion of a person year to development work and some, like Kidd Creek Mines Ltd. may go as high as ten.

Consultants are not usually involved in R & D, but some such as Bacon and Donaldson from Vancouver will perform actual research on base metal projects.

Recent Past and Ongoing R & D

Past, present and possible future R & D can best be summarized as follows:

YEAR	DESCRIPTION	PERFORMER	FUNDING
Bulk Base Metal Concentrates			
1975-83	Sulphation-roast-leach process for bulk concentrates	Research Productivity Council - New Brunswick	$800K DREE $250K Anaconda
1976-83	Modified pressure sulphuric acid leach process for bulk concentrates	Sherritt Gordon	$300K DREE $120K Sherritt $120K CANMET
1976-83	Ferric chloride leach process and dry way chlorination leach process for bulk concentrates	CANMET	$4500K
1970-83	Chloride leach processes for	USBM	

YEAR	DESCRIPTION	PERFORMER	FUNDING
1975-83	bulk concentrates	Elkem (Norway)	
1973-83		Minemet (France)	
Lead Concentrates (Hydrometallurgical Research)			
1975-83	Chloride leach/fused salt electrolysis process	USBM	$50K/a (USBM, St.Joe Minerals, Asarco, and Cominco)
1975-77		Cominco	$500K Cominco
Pyrometallurgical Research (Oxygen Smelting)			
1970-82	KIVCET Process	Russia	
1977-83	QSL Process	Germany	
1979-83	Outokumpu Process	Finland	
Copper and Nickel Concentrates			
Pyrometallurgical Research			
1980-83	New roast-electric smelt process for nickel	Inco	$20 000K
1978-82	Continuous smelting-converting Inco process for nickel and copper		$5000K (50% Inco; 50% IT&C)
1982-83	SO_2 capture and conversion to magnesium sulphate	SNA/Noranda	$1000K (25% Noranda; 25% SNA; 50% Quebec Gov't.
1981-82	Basic studies on slag-matte-metal systems	CANMET	2 PY's
Hydrometallurgical Research			
1979-82	Ferric chloride leach process for copper	Bacon/Donaldson Co. Ltd.	$800K (Great Central Mines)
1975-83	Pressure leach process in sulphate/chloride media	Noranda	
1981-82	Review of hydrometallurgical process for nickel and copper	CANMET	1 PY

YEAR	DESCRIPTION	PERFORMER	FUNDING
1970-82	Chloride leach processes for copper	Cyprus Metallurgical (USA)	$20 000K
1975-83	Chloride leach processes for copper	Elkem (Norway)	
1970-83	Chloride leach processes for copper	Minemet (France)	
1972-83	Chloride leach processes for copper	Australia	
1981-83	Pressure leach-hydrogen reduction process for copper	UBC	NSERC

GAPS AND CHALLENGES

Because of the diversity of commodities and processes required to produce metals from ores and the general review of problems facing the industry, as outlined, the gaps and challenges confronting the industry are grouped and profiled under beneficiation and smelting.

BENEFICIATION

Existing concentration techniques are not entirely adequate for many commodities. New techniques and more selective promoters are needed to increase the recovery of nickel, cobalt, platinum, molybdenum. Reducing the pyrrhotite content in nickel concentrates would improve SO_2 emission problems. Sulphide slimes represent a major loss of metal values in tailings, sometimes as high as two thirds of the total loss. Slimes of other minerals have been successfully concentrated by selective flocculation (hematite) or piggy-back flotation (anatase from kaolinite using calcite as a carrier) but so far no one has been successful in applying these techniques to sulphide slimes.

Byproduct recovery from tailings leaves much to be desired. There are still no truly effective techniques for recovering the small but very high value contents of Sn, Co, Bi in sulphide flotation plant mill tailings.

A prime example of loss of mineral resources is tin in the Kidd Creek mill tailings. The tailings contain 0.13% Sn which amounts to about 10 t/d or 3500 t/a. At $6.25/lb Canadian ($5.00 US) yearly value is $43.75 million. State-of-the-art gravity concentration only recovers about 10% of the tin in the tailings. Tin content in the Brunswick Mining and Smelting (BMS) Corporation and Heath Steel (HS) mill tailings is reported to be 0.107% and over 0.0905% respectively (7, 81). Assuming 7700 t/d BMS tailings and 3500 t/d HS tailings, yearly loss in the two tailings is estimated at 3900 t or $48.75 million.

Other valuable byproducts which are recovered at some stage of the principal base metal beneficiation process, e.g., with zinc: cadmium, germanium, indium, gallium; with lead: bismuth; with copper: selenium, tellurium, are lost through shipment of concentrates to overseas smelters. The shipper receives no payment for the minor amounts of valuable elements contained in the concentrates. Therefore, as far as Canada is concerned, this is a loss of mineral resources and methods must be devised to selectively beneficiate these values.

CANMET should initiate a program of gold and silver recovery from Canadian ores and tailings by heap leach cyanidation, carbon-in-pulp cyanidation and electrowinning. The cyanidation of gold ores is in a rebirth stage due to a multitude of innovations (82). CANMET should gain more knowledge on this new technology in order to transfer information to both large and small Canadian developers. A definite technological gap exists in this area, which may be classified as high risk and long term. Canadians interested in this technology have had to go south of the border where the technology is in industrial application in Nevada and the Western States. The advantages of heap leaching are:

1. Capital and operating costs are 20% and 40% respectively of conventional milling.
2. Minimum surface disturbance.
3. Less waste generation or land restoration.
4. Shorter lead time of 2 to 3 years before production compared with 7 to 10 years for conventional mining.

Heap leach cyanidation may be the process to increase extraction efficiency of the silver by-product in the treatment of base-metal ores.

Another problem in gold processing requiring attention is the arsenopyrite present in numerous small deposits which cannot be exploited because there is not enough roasting capacity in existing facilities. Novel processes to circumvent roasting, which is not environmentally desirable, would be of major value.

Not enough of nature's available tools have been made use of or exploited enough in the form of bacteria to alter surfaces, remove impurities or unacceptable rejects, of vegetation which has affinities for definite species of material and which will selectively concentrate certain elements in their cell structure. Research in these areas will create very pleasant surprises.

To alleviate costs in mining, materials handling, equipment buildings, reagents, preconcentration techniques and equipment must be rethought, refined and incorporated in flow sheet designs. To maximize product values, properties of certain types of ores must be taken advantage of to obtain superconcentrates which can compete with chemically produced materials and thus command higher prices.

Two major areas of research which are somehow interrelated but require wide ranging experimentation and research to produce benefits in the next 20 years are fine particle processing and super selective surface reagents. Conservation of resources requires 100% recoveries and thus elimination of losses due to fines and slimes; complete utilization of all our resources requires treatment of fine complex ore bodies. What basically needs to be better understood and translated into practice are the surface chemical properties of minerals and the means to make these characteristics more prevalent than the bulk properties of the material through an enhancement by more selective promoters and separation techniques designed specifically for that purpose. Equipment will be required to translate these findings into practical applications. Energy considerations must be taken into account as liberation is a very important factor.

Mineral liberation must be considered separately as it is of such high magnitude; in most cases it is the most expensive part of concentration. New concepts must be investigated to move away from the age-old inefficient comminution schemes. This avenue of research has been promoted many times and for a long time, but the problem still exists. What can be solved within a reasonable length of time is the development of on-line liberation probes to obtain optimal control and automation of comminution circuits. The instrumentation to measure liberation is already developed for batch samples. Its impact on process understanding and optimization can be seen and the concept of liberation is quickly overriding that of size characterization.

Effective controls and instrumentation in mineral dressing plants are still to be achieved. This relatively slow development is caused by the still-to-be determined operating variables and the harsh environment in which the instrumentation must function. Modelling and simulation of unit operations and flow sheets will determine specifically where controls will be necessary. Instruments will then have to be devised to permit these controls to be effective. Innovative research will be necessary to devise the types of probes necessary.

Many unit operations will face greater challenge in the coming years and new separation concepts must be thoroughly explored to offer alternatives to present beneficiation techniques. Most of these are in the field of fine particles. Of importance are solid-liquid separation, materials handling facilities, high gradient magnetic separation, magnetohydrostatic and magnetohydrodynamic systems, cryogenic systems, flocculation-flotation systems and magneto-cyclones. It is crucial to delve deeply in this area and effect a commendable technology transfer to our mineral industry. Moreover, processing in "space" should be considered and this research will have to deal with entirely new problems, i.e., low or zero gravity, no air or water, low atmospheric pressure or vacuum and extreme temperatures.

Mining in the far North is hampered by many constraints and processing problems, but for Canada to reach its full potential, a comprehensive long range approach must be realized to solve barriers which are impeding progress. A large number of research projects must be undertaken, some mentioned in topics already discussed, but which must be directed towards conditions in the North - flotation reagents which will perform selectively in cold water, more effective filtration, modular transportable mills, greater use of automation and controls, total recycling of process water, greater regard for a fragile ecology, etc.

Energy conservation has become a perennial topic for discussion and so it should be. Again many of the topics mentioned include areas where new equipment, new instrumentation and controls, new techniques, new methods, preconcentration strategies will offer great possibilities for energy savings. Particular attention must be given to a project to improve separation efficiencies in Canadian mineral processing plants which is long range but will produce extremely valuable information and solutions to liberation problems and alternative flotation schemes.

Decades will permit us to win a number of battles against human and environmental pollution but it must continue to be an all out war and must resolve problems dealing with dust, toxic fumes and compounds, noise, radiation, release of dangerous elements in the ground and water systems through tailings disposal. Future research must concentrate on elimination of dust through surface active suppressants, abatement of noise through better equipment design, 100% recirculation systems, new concentration flow sheets to remove deleterious elements, and improved individual protection equipment.

SMELTING/REFINING

There is little doubt that the control of SO_2 emissions will be the major problem facing Canada's base metal smelting industry over the next decade. However there are other factors to consider which have a very significant effect on the viability and competitiveness of the smelting/refining process, such as energy efficiency, impurity elimination, byproduct marketability, major metal and byproduct metals recovery and capital and operating costs. Many of these may be adversely affected by the measures taken to facilitate SO_2 control. The ideal process technology therefore may have to be a compromise which will minimize environmentally related problems and obtain acceptable recoveries from the resource base, while maintaining the long term competitiveness of Canda's base metal smelting industry.

COPPER/NICKEL

SMELTING/CONVERTING

The SO_2 problem is more severe for copper and nickel than it is for zinc and lead because of the high sulphur to metal ratio in the concentrates. If all of the sulphur in the concentrates is converted to sulphuric acid the resulting acid production will amount to about 3 t/t of copper and nearly 9 t/t of nickel. In many developed countries, particularly Japan, very high recoveries of SO_2 from smelter gases have been achieved using existing technology. This has been accomplished by blending converter gases with flash smelter gases for conversion of SO_2 to acid, by producing saleable gypsum from dilute SO_2 gas streams and by using expensive SO_2 absorption processes (83,84,85) to concentrate the SO_2 and facilitate conversion to acid or gypsum. However these methods are economically viable only if markets for the byproducts are very close. In Canada this is not the case. Markets for byproduct gypsum do not exist, and the netback to Canadian smelters from acid sales seldom comes close to covering the cost of production. The problem is particularly severe for smelters in isolated inland locations where transportation costs to markets can be higher than the value of the acid. Although liquid SO_2 from the Inco flash smelter commands a premium price relative to acid, markets are extremely limited, and sales seldom match production capacity. Even if a viable continuous smelting/converting process such as the Mitsubishi copper process was available for both copper and nickel to facilitate SO_2 recovery, the problems of marketing acid or liquid SO_2 would still be paramount. Canada's SO_2 problem therefore is unique and requires unique solutions particularly for copper/nickel. It is necessary to develop new uses and/or other alternatives to improve the marketability of the sulphur byproduct by converting SO_2 at the smelter site to other higher value saleable products such as fertilizer, elemental sulphur or sulphur chemicals.

Those processes which make the most efficient use of pure oxygen (such as Inco's flash smelter) to produce very high SO_2 concentrations should facilitate the economic conversion of SO_2 to elemental sulphur or other sulphur chemicals. These technologies, designed to minimize the cost of SO_2 control, can be applied to all pyrometallurgical processes which produce SO_2 as a byproduct regardless of the process or the commodity. They are therefore extremely important, not only to provide acceptable solutions to the "acid rain" problem, but also to the long term competitiveness of Canada's base metal smelting industry.

To facilitate the introduction of continuous energy efficient smelting/converting technology for both nickel and copper it is desirable to minimize the problems related to impurity elimination and metal and byproduct recovery. This requires the development of a much more sophisticated database on the chemical and physical properties of molten slag-matte-metal systems, and the effect of gas phase composition on their equilibria composition. The development of better methods to reject undesirable impurities such as arsenic, antimony and bismuth, and to reduce copper and precious metal losses to the slag will greatly enhance the viability and flexibility of continuous processes for copper. The recovery of molybdenum and cobalt should also be investigated where significant quantities of these elements are present. For continuous nickel processes, impurities of some concern are arsenic, bismuth, lead and selenium, while the major effort in improving recovery should be directed toward cobalt and platinum group metals, as well as nickel and copper. Because of the very high iron content in nickel concentrates, very small reductions in the metals content of the slag can result in significant increases in recovery.

Efforts should also be continued to improve the energy efficiency of pyrometallurgical processes for nickel. As with copper this can be best accomplished by maximizing the utilization of the sulphur fuel in the concentrate, and by developing a continuous counter-current process to produce high-grade matte and discard slag.

REFINING

Although refining processes for copper and nickel are well established and have few problems, research directed toward obtaining a better understanding of the factors controlling impurity disposition in copper and nickel electro-refining and electrowinning cells would provide guidance with respect to the degree of impurity removal required in the smelting/converting processes. This could also lead to improved recovery methods, and ultimately to improved recovery of byproducts, such as cobalt. Electrolysis R & D from chloride media is particularly relevant for nickel, since many of the refining processes already involve chloride solutions for both electrowinning and electrorefining. However, hydrometallurgical processes for either nickel or copper are also likely to be carried out in chloride media, thus making electrowinning studies relevant to both metals.

HYDROMETALLURGICAL PROCESSES AND OTHER ALTERNATIVES TO SMELTING

Hydrometallurgical processes do not produce SO_2 and therefore have virtually no environmental constraints or production constraints related to sulphuric acid marketability. Some of the processes can also convert most of the sulphides in base metal concentrates to elemental sulphur, which can be recovered and sold if markets are available, or stored indefinitely with no detrimental environmental effects. Elemental sulphur is also a much higher value commodity than sulphuric acid and can be shipped at least three times as far for the same cost. Sulphur producing processes therefore could be ideal for inland base metal sulphide mines located far from acid markets. They can however have other problems related to the control of aqueous emissions, precious metals recovery, energy requirements or iron removal.

For copper the most desirable processes are those that convert sulphides in the concentrates directly to elemental sulphur, while maintaining copper and iron in their lower valence states. This minimizes the capital and energy costs of the process, and can be accomplished effectively only if the process is carried out in chloride media. Iron rejection is also facilitated in chloride media and a saleable iron product can be produced if economics so warrant. However, precious metals recovery and the production of wire bar grade copper remains a problem. R & D is required to develop adequate methods for silver removal and recovery from chloride leach liquors to facilitate the production of wire bar grade copper; to optimize the design of diaphragm electrolytic cells for copper chloride electrowinning, and to recover gold, sulphur and molybdenum from leach residues. Research to facilitate the recovery of a sponge iron byproduct rather than a discard iron residue would also be useful to maximize resource utilization, and possibly improve the process economics.

For nickel, the most desirable alternative process to conventional smelting would also be one that converts sulphides in the concentrate to elemental sulphur. However, as stated earlier nickel concentrates contain more iron than copper concentrates, and are usually much more complex, containing copper, cobalt, and precious and platinum group metals. To ensure the economic viability of the process, these valuable byproducts must be recovered, and iron must be removed in an economically and environmentally acceptable manner. None of the processes developed to date can efficiently recover nickel and associated byproducts from nickel concentrates while converting sulphides to elemental sulphur. As with copper the most promising processes appear to be those that utilize chloride metallurgy to facilitate conversion of sulphides to elemental sulphur and iron to insoluble oxides. Because of the complexity of nickel concentrates a great deal more basic and applied research is required to optimize process operating parameters for rejecting iron; converting sulphides to sulphur; recovering precious and platinum group metals and recovering pure nickel, copper and cobalt products from chloride leach solutions. To accomplish this it is not necessary to rely exclusively on hydrometallurgy. A pyrometallurgical extraction process, involving dry chlorination/oxidation of sulphide concentrates (52,70,86) can be very effectively utilized to convert sulphides to elemental sulphur, iron to insoluble hematite and non-ferrous metals to soluble chlorides, which can then be separated and refined by hydrometallurgical techniques. The pyrometallurgical steps in the process can be exceptionally energy efficient, and the process has the potential to recover all of the valuable components in the concentrate, including sulphur and iron, as saleable products. The process therefore could be ideal for conventional nickel sulphide concentrates or nickeliferrous pyrrhotite, particularly in Canadian inland locations where SO_2 and sulphuric acid marketing problems pose a considerable constraint to further processing of these ore deposits. Sulphur and possibly iron could also add substantially to the total revenues, making process economics more attractive. However, the dry chlorination/oxidation process has not been tested on nickel concentrates, and a substantial long term R & D effort is required to demonstrate the technical and economic viability of the technology and its applicability to Canadian ore deposits.

ZINC/LEAD

ZINC CONCENTRATES

For conventional zinc concentrates the control of SO_2 emissions is not a serious problem. In the roast leach electrowin (RLE) process, SO_2 from the fluid bed roasters is readily amenable to conversion to H_2SO_4 in an acid plant. If acid marketing is a problem the Sherritt pressure sulphuric acid leach (PSA) process can be used to produce sulphur instead of SO_2 and acid. Neither process has any serious problems in treating high grade, relatively pure zinc concentrates. However zinc concentrates containing appreciable amounts of silver can pose a problem. Recovery of silver from RLE process leach residues is often difficult because of the complexity of the jarosite circuit for iron removal. In the PSA process, silver recovery is complicated due to the presence of sulphur and pyrite in the leach residues. Research is required to obtain a better understanding of the factors controlling silver solubility and silver disposition in the leach circuits, which will lead to the development of improved methods for silver recovery.

BULK CONCENTRATES

In many cases, ore deposits are too complex and fine grained to allow the efficient production of high grade zinc, lead and copper concentrates by conventional beneficiation techniques. Under such circumstances a bulk concentrate must be produced to obtain acceptable recoveries from the ore body. Bulk concentrates can contain any two or more of the base metals zinc, lead and copper, plus silver, gold and other associated byproducts, and are not amenable to treatment in the conventional RLE or PSA process. The Imperial Smelting Process (ISP) does not have the flexibility to handle the range of metal ratios and grades encountered in the bulk concentrates. It is necessary therefore to develop alternative technologies which will efficiently and economically recover the valuable metals from these complex relatively low grade concentrates while minimizing associated environmental problems. The RPC Sulphation Roast Leach (SRL) process (65) and the Sherritt Gordon Modified Pressure Sulphuric Acid (MPSA) process (66) are two Canadian technologies which have an excellent chance of being technically and economically viable for application to bulk concentrates in the short to medium term. As with zinc concentrates the two processes provide industry with the flexibility to produce either acid (SRL process) or elemental sulphur (MPSA process) as a saleable byproduct. The SRL process can also be applied directly to the complex sulphide ore, if such an option should prove to be more attractive than producing a bulk concentrate. However, considerably more development work is required on a pilot scale on both the SRL and MPSA processes, particularly with respect to improving precious metals recovery, minimizing the costs of producing a high value lead product and eliminating problems associated with roaster gas cooling and cleaning (SRL process) and sulphur recovery (MPSA process).

As with copper and nickel, processes which employ chloride metallurgy to convert sulphides to sulphur can also be attractive options in the medium to long term for bulk concentrates. The hydrometallurgical chloride leach processes (25,67,68,69), all of which can be carried out at atmospheric pressure, could be attractive alternatives to the Sherritt MPSA process. The dry chlorination/oxidation pyrometallurgical process (70) could be ideal for highly pyritic ores or bulk concentrates, particularly in cases where elemental sulphur and possibly iron can provide substantially more revenues than could be obtained from sulphuric acid sales (SRL process).

Both the hydro and pyro chloride metallurgy alternatives require substantially more R & D to demonstrate the technical and economic viability of the technology. The dry chlorination process in particular requires laboratory and pilot scale studies to verify design criteria for flash chlorination and for the gaseous conversion of ferric chloride to hematite. Both hydro and pyro processes require research to optimize solution purification and electrolysis techniques in chloride media for both zinc and lead, with fused salt electrolysis being particularly important for the latter.

LEAD CONCENTRATES

Because of health, safety and energy efficiency considerations, the conventional pyrometallurgical sinter/blast furnace process for lead concentrates is almost certain to be replaced by a new generation lead smelter which will employ flash smelting/slag cleaning techniques designed to maximize the use of the sulphur energy in the concentrate and obtain high recoveries of lead and associated byproducts, while minimizing health and environmental problems due to materials handling and fugitive emissions. The process will also have to be capable of handling a variety of lead concentrates and lead residues containing varying levels of undesirable impurities. Regardless of the pyrometallurgical process the development of a better database on the chemical and physical properties of molten slag-metal systems and the deportment of impurities will greatly assist the process developers to reduce metal losses to the slag and increase the flexibility of the process to recover byproducts and reject undesirable impurities. R & D should also be directed toward obtaining a better understanding of the factors controlling impurity disposition in lead electrorefining cells, which would provide guidance for the degree of impurity removal required in the smelting process. The effect on the recovery of byproducts such as copper, antimony and bismuth, would also have to be investigated in cases where thermal refining techniques must be replaced by electro-refining to meet work place emission regulations.

From an environmental and health safety point of view the hydrometallurgical ferric chloride leach process for lead concentrates is probably ideal since it operates at low temperatures and produces sulphur instead of SO_2. Lead extraction and recovery is also very high. However the recovery of precious metals and other byproducts, and the ability of the process to handle a variety of lead concentrates and residues has not been adequately investigated. Much more R & D is also required on a bench and pilot plant scale to optimize cell design for fused salt electrolysis of lead chloride, and to determine the technical and economic advantages of this approach versus aqueous electrolysis, for the recovery of lead and associated byproducts.

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APPENDIX - DIRECTORY OF FEDERALLY SUPPORTED RESEARCH IN UNIVERSITIES

1980-1981

1. Nickel recovery fom laterite acid leach solutions by precipitate flotation. 2. Metal precipitation from loaded organic extractants by reaction with hydrogen; Distin, P.A.; Mining and metallurgical engineering; McGill University, Montreal, Que.; Natural Sciences and Engineering Research Council; $9042

The molecular basis for the toxicity of hydrogen sulphide; Donini, J.C.; Chemistry; St. Francis Xavier University, Antigonish, N.S.; Natural Sciences and Engineering Research Council; $10 000

The molecular basis for the toxicity of hydrogen sulphide; Donini, J.C.; Chemistry; St. Francis Xavier University, Antigonish, N.S.; Natural Sciences and Engineering Research Council; $25 000

Origin and metallogenic relationships of massive base metal sulphide and gold deposits; Hutchinson, R.W.; Geology; University of Western Ontario, London, Ont.; Natural Sciences and Engineering Research Council; $18 637

Computer analysis of phase diagrams and thermodynamic properties of molten salt, oxide, silicate and sulphide systems; Lin, P.L.; Metallurgical Engineering; Ecole Polytechnique, Montreal, Que.; Natural Sciences and Engineering Research Council; $8000

Phase relations and field studies pertaining to magmatic and massive volcanogenic sulphide ore deposits; Maclean, W.H.; Geological sciences; McGill University, Montreal, Que.; Natural Sciences and Engineering Research Council; $9500

Geology and geochemistry of massive sulphide ores; Scott, S.D.; Geology; University of Toronto, Ont.; Natural Sciences and Engineering Research Council; $21 750

Metal oxidation, ionic diffusion and defect structures in solid metal oxides, nitrides and sulphides; Smeltzer, W.W.; Metallurgy/Materials Science; McMaster University, Hamilton, Ont.; Natural Sciences and Engineering Research Council; $33 500

1979-1980

1. Nickel recovery from laterite acid leach solutions by precipitate flotation. 2. Metal precipitation from loaded organic extractants by reaction with hydrogen; Distin, P.A.; Mining and Metallurgical Engineering; McGill University; Montreal, Que.; Natural Sciences and Engineering Research Council; $8235

The molecular basis for the toxicity of hydrogen sulphide; Donini, J.C.; Chemistry; St. Francis Xavier University, Antigonish, N.S.; Natural Sciences and Engineering Research Council; $9400

The molecular basis for the toxicity of hydrogen sulphide; Donini, J.C.; Chemistry; St. Francis Xavier University, Antigonish, N.S.; Natural Sciences and Engineering Research Council; $16 560

Recovery of metal values from complex sulphide ores; Habashi, F.; Mining/Metallurgy; Université Laval, Québec, Qué.; Energy, Mines and Resources, Canada; $9000

Origin and metallogenic relationships of massive base metal sulphide and gold deposits; Hutchinson, R.W.; Geology; University of Western Ontario, London, Ont.; Natural Sciences and Engineering Research Council; $16 974

Phase relations and field studies pertaining to magmatic and massive volcanogenic sulphide ore deposits; Maclean, W.H.; Geological sciences; McGill University, Montreal, Que.; Natural Sciences and Engineering Research Council; $11 136

Platinum group elements and other trace elements in magmatic Ni-Cu sulphide ores associated with different host rocks; Naldrett, A.J.; Geology; University of Toronto, Ont.; Energy, Mines and Resources, Canada; $13 500

Oxidation kinetics and the beneficiation of fine particle metal sulphides; Schreiber, H.P.; Chemical Engineering; Ecole Polytechnique, Montreal, Que.; Energy, Mines and Resources Canada; $5000

Petrology and geochemistry of sulphide ores and minerals; Scott, S.D.; Geology; University of Toronto, Ont.; Natural Sciences and Engineering Research Council; $17 718

Metal oxidation, ionic diffusion and defect structures in solid metal oxides, nitrides and sulphides; Smeltzer, W.W.; Metallurgy/Materials Science; McMaster University; Hamilton, Ont.; Natural Sciences and Engineering Research Council; $32 913

1978-1979

1. Physical chemistry of hydrometallurgical reactions. 2. Thermodynamics and kinetics of reactions involving molten metal sulphides. 3. Application of electrochemical processes to inorganic pollution; Byerley, J.J.; Chemical Engineering; University of Waterloo, Ont.; Natural Sciences and Engineering Research Council; $15 339

The molecular basis for the toxicity of hydrogen sulphide; Donini, J.C.; Chemistry; St. Francis Xavier University, Antigonish, N.S.; Natural Sciences and Engineering Research Council; $16 000

Origin and metallogenic relationships of massive base metal sulphide and gold deposits; Hutchinson, R.W.; Geology; University of Western Ontario, London, Ont.; Natural Sciences and Engineering Research Council; $16 400

Phase relations and field studies pertaining to magmatic and massive volcanogenic sulphide ore deposits; Maclean, W.H.; Geological Sciences; McGill University, Montreal, Que.; Natural Sciences and Engineering Research Council; $10 759

Platinum group elements and other trace elements in magmatic Ni-Cu sulphide ores associated with different host rocks; Naldrett, A.J.; Chou, C.L.; Hoffman, E.L.; Geology; University of Toronto, Ont.; Energy, Mines and Resources Canada; $11 500

Oxidation kinetics and the beneficiation of fine-particle metal sulphides; Schreiber, H.P.; Chemical Engineering; Ecole Polytechnique, Montreal, Que.; Energy, Mines and Resources Canada; $6000

Petrology and geochemistry of sulphide ores and minerals; Scott, S.D.; Geology; University of Toronto, Ont.; Natural Sciences and Engineering Research Council; $17 119

Metal oxidation, ionic diffusion and defect structures in solid metal oxides, nitrides and sulphides; Smeltzer, W.W.; Metallurgy/Materials Science; McMaster University, Hamilton, Ont.; Natural Sciences and Engineering Research Council; $31 800

Ni-Cu-Fe sulphide-carbonate-silicate relationships in altered ultramafic rocks and their host volcanic rocks; Watkinson, D.H.; Geology; Carleton University, Ottawa; Ont.; Energy, Mines and Resources Canada; $1350

1977-1978

1. Physical chemistry of hydrometallurgical reactions. 2. Thermodynamics and kinetics of reactions involving molten processes to inorganic pollution; Byerley, J.J.; Chemical Engineering; University of Waterloo, Ont.; National Research Council of Canada; $14 471

Origin and metallogenic relationships of mineral deposits: expecially of precambrian nickel and base metal sulphide, gold, iron and pegmatite deposits; Hutchinson, R.W.; Geology; University of Western Ontario, London, Ont.; National Research Council of Canada; $13 550

Phase relations and field studies pertaining to magmatic and massive volcanogenic sulphide ore deposits; Maclean, W.H.; Geological Sciences; McGill University, Montreal, Que.; National Research Council of Canada; $10 150

Platinum group elements and other trace elements in magmatic Ni-Cu sulphide ores associated with different host rocks; Naldrett, A.J., Chou, C.L., Hoffman, E.L.; Geology; University of Toronto, Ont.; Energy, Mines and Resources Canada; $8000

Petrology and geochemistry of sulphide ores and minerals; Scott, S.D.; Geology; University of Toronto, Ont.; National Research Council of Canada; $16 150

Metal oxidation, ionic diffusion and defect structures in solid metal oxides, nitrides and sulphides; Smeltzer, W.W.; Metallurgy/Materials Science; McMaster University, Hamilton, Ont.; National Research Council of Canada; $30 000

Oxidation kinetics and the beneficiation of fine-particle metal sulphides; Schreiber, H.P.; Chemical Engineering; Ecole Polytechnique, Montreal, Que.; Energy, Mines and Resources Canada; $8500

1976-1977

1. Physical chemistry of hydrometallurgical reactions. 2. Thermodynamics and kinetics of reactions involving molten metal sulphides. 3. Application of electrochemical processes to inorganic pollution; Byerley, J.J.; Chemical Engineering; University of Waterloo, Ont.; National Research Council of Canada; $13 350

Study for cooperative research on modelling of grinding and classification for fine-grained sulphide ores of New Brunswick. On behalf of EMR Canada; Everell, M.D.; Mines; Université Laval, Québec, Qué.; Supply and Services Canada; $18 117

1. Minéralisation et problemes d'exploitation de gisements dans le region de Chibougamau. 2. Etude de la minéralisation Pb-Zn dans le bassin de Mistassini; Guh, J.; Sciences appliquées; Université du Québec à Chicoutimi, Qué.; Conseil National de Recherches du Canada; $6903

Origin and metallogenic relationships of mineral deposits; especially of precambrian nickel and base metal sulphide, gold, iron and pegmatite deposits; Hutchinson, R.W.; Geology; University of Western Ontario, London, Ont.; National Research Council of Canada; $12 500

Platinum group elements and other trace elements in magmatic Ni-Cu sulphide ores associated with different host rocks; Naldrett, A.J.; Geology; University of Toronto, Ont.; Energy, Mines and Resources Canada; $8500

1. Binary mineral flotability in the presence os arsenic trioxide 2. Dithiophosphate collectors and their adsorption characteristics on nickel sulphide minerals 3. The mechanism of copper activiation of sphalerite; Salman, T.M.; Mining/Metallury; McGill University, Montreal, Que.; National Research Council of Canada; $10 850

Physical geochemistry of sulphide ores; Scott, S.D.; Geology; University of Toronto, Ont.; National Research Council of Canada; $12 086

Ni-Cu-Fe-sulphide-carbonate-silicate relationships in altered ultramafic rocks; Watkinson, D.H.; Geology; Carleton University, Ottawa, Ont.; Energy, Mines and Resources Canada; $7500

1975-1976

Oxidation of hydrogen sulphide to elemental sulphur on activated carbon; Andersen, N.E.; Chemical Engineering; University of Calgary, Alta.; National Research Council of Canada; $2492

1. Physical chemistry of hydrometallurgical reactions. 2. Thermodynamics and kinetics of reactions involving molten metal sulphides. 3. Application of electrochemical processes to inorganic pollution; Byerley, J.J.; Chemical Engineering; University of Waterloo, Ont.; National Research Council of Canada; $11 410

Pedologic inventory of mining and milling wastes in specified areas of Quebec (as a member of the cooperative "B" level pit slope stability project 1972-77 - revegetation). On behalf of EMR Canada; Cossette, M.; Environment; Université de Sherbrooke, Que.; Supply and Services Canada; $30 000

1. Minéralisation et problemes d'exploitation de gisements dans la région de Chibougamau. 2. Etude de la minéralisation Pg-Zn dans le bassin de Mistassini; Guha, J.; Sciences appliquées; Université du Québec à Chicoutimi, Qué.; Conseil national de recherches du Canada; $6730

Origin and metallogenic relationships of mineral deposits: especially of precambrian nickel and base metal sulphide, gold, iron and pegmatite deposits; Hutchinson, R.W.; Geology; University of Western Ontario, London, Ont.; National Research Council of Canada; $12 190

Pedologic inventory of sulphide mine wastes of Endako Mine at Fraser Lake, Taxada Mine of Texada Island and Coast Copper on Vancouver Island (as a member of the cooperative "B" level pit slope project. On behalf of EMR Canada; Lavkulich, L.M.; Soil Science; University of British Columbia, Vancouver, B.C.; Supply and Services Canada; $26 620

Development of solvent extraction system for cobalt and nickel in acid media; Lever, A.B.P.; Walker; I.M.; Chemistry; York University, Downsview, Ont.; Energy, Mines and Resources Canada; $9000

1974-1975

Oxidation of hydrogen sulphide to elemental sulphur on activated carbon; Andersen, N.E.; Chemical Engineering; University of Calgary, Alta.; National Research Council of Canada; $4300

1. Physical chemistry of hydrometallurgical reactions. 2. Thermodynamics and kinetics of reactions involving molten metal sulphides. 3. Application of electrochemical processes to inorganic pollution. Byerley, J.J.; Chemical Engineering; University of Waterloo, Ont.; National Research Council of Canada; $10 000

1. Minéralisation et problemes d'exploitation de gisements dans la région de Chibougamau. 2. Etude de la minéralisation Pb-An dans le bassin de Mistassini; Guha, J.; Sciences Appliquées; Université du Québec à Chicoutimi, Qué.; Conseil national de recherches du Canada; $5900

Development of solvent extraction system for cobalt and nickel in acid media; Lever, A.B.P.; Walker, I.M.; Chemistry; York University, Downsview, Ont.; Energy, Mines and Resources Canada; $10 000

Physical geochemistry of sulphide ores; Scott, S.D.; Geology; University of Toronto, Ont.; National Research Council of Canada; $11 785; Physical geochemistry of sulphide ores; Scott, S.D.; Geology; University of Toronto, Ont.; National Research Council of Canada; $10 330

1. Volcanogenic sulphide deposits in the Springdale ophiolite. 2. Comparison of volcanogenic deposits in different tectonic settings. 3. Petrochemistry and ore bearing potential of Newfoundland ultramafic rocks. 4. Geothermometry and sulphide isotope composition of fluorite veins at St. Lawrence and of barite-celestite-galena deposits in carboniferous rocks of western Newfoundland; Smitheringale, W.G.; Geology; Memorial University of Newfoundland, St. John's, Nfld.; National Research Council of Canada; $2975

1. Radiation chemistry of aqueous solutions of sulfhydryl compounds 2. Gas phase ionic reactions in hydrocarbon hydrogen sulphide mixtures; Stone, J.A.; Chemistry; Queen's University, Kingston, Ont.; National Research Council of Canada; $11 000

1973-1974

1. Physical chemistry of hydrometallurgical reactions. 2. Thermodynamics and kinetics of reactions involving molten metal sulphides. 3. Application of electrochemical processes to inorganic pollution; Byerley, J.J.; Chemical Engineering; University of Waterloo, Ont.; National Research Council of Canada; $10 000

Development of solvent extraction system for cobalt and nickel in acid media; Lever, A.B.P., Walker, I.M.; Chemistry; York University, Downsview, Ont.; Energy, Mines and Resources Canada; $10 000

Geochemical relations of sulphide-silicate-oxide assemblages; Macrae, N.D.; Geology; the University of Western Ontario, London, Ont.; National Research Council of Canada; $4850

1972-1973

Geochemical relations of silicate-oxide-sulphide assemblages; Macrae, N.D.; Geology; University of Western Ontario, London, Ont.; National Research Council of Canada; $5400

1. Geological and geochronological studies in S. Norway 2. Studies of U-Ag-Cu-Bi Mineralization in the N.W.T. and of mineralization in N. Saskatchewan; Morton, R.D.; Geology; University of Alberta, Edmonton, Alta.; National Research Council of Canada; $7500

1971-1972

1. Fin - les minerais nickelifères de Mattagami Lake Mines. 2. Etudes de minerais nikelifères de Dumont Nickel Ltd.-Abitibi; Assad, J.R.; Géologie; Université Laval, Québec, Qué.; La commission géologique du Canada, EMR; $1255

Contribution de la microsonde électronique à l'étude de l'altération minérale: géochimie du Cr, Mn, Fe, Ni, Co et Cu; Coy-Yll, R.; Génie Géologique; École Polytechnique, Montréal, Qué.; Conseil National de recherches du Canada; $7000

Geochemical relations of silicate-oxide-sulphide assemblages; Macrae, N.D.; Geology; University of Western Ontario, London, Ont.; National Research Council of Canada; $4000

1. Geological and geochronological studies in S. Norway. 2. Studies of U-Ag-Cu-Bi mineralization in the N.W.T. and of mineralization in N. Saskatchewan; Morton, R.D.; Geology; University of Alberta, Edmonton, Alta.; National Research Council of Canada; $7500

IRON AND STEEL

GENERAL DESCRIPTION OF INDUSTRY

Canada is a major producer of iron ore, supplying 50 Mt of usable ore, valued at approximately $1800 million in 1980, representing about 6% of total world production. Within the Canadian metals industry, iron ore production is a close second to copper and accounts for 19% of total value of metals compared with 20% for copper. Based on 1980 figures, the major producing areas are (1):

Province	Tonnes x 1000, natural weight
Quebec-Labrador	42 576
Ontario	6 550
British Columbia	635
	49 761

As shown, the Quebec-Labrador area is the principal producer with about 85% of the Canadian output (1). The Canadian industry is largely export-oriented and foreign-owned with the USA controlling 80% of production (1).

Although the iron and steel industry is still essentially a supplier of domestic needs, there has been a recent shift from a small net importer of steel to a small exporter. Annual crude steel production totalled about 16 Mt in 1980 with about 3 Mt being exported. The Canadian steel industry is largely Canadian-owned comprising about 24 plants. Five companies (Sysco, Dofasco, Stelco, Algoma and Sidbec-Dosco) are iron and steel producers. These plants are located mainly along the Great Lakes/St. Lawrence Seaway system (2).

An important requirement of the steel industry is the ready availability of ferroalloys, which are additives for modifying the properties of steel. Ferroalloys of chromium, manganese and silicon are produced world wide in large tonnages, while ferroalloys of columbium, (niobium), vanadium, molybdenum and tungsten are produced in relatively small volumes.

Canada depends on imports to meet its chromium, manganese and vanadium requirements. Although Canada is capable of producing all of its ferroalloy needs, the economics dictate otherwise. The industry is mostly foreign-controlled and relies on domestic supply in a branch-plant type arrangement. The total value of Canadian consumption in 1980 was approximately $135 million (3).

CURRENT TECHNOLOGY

The demands of modern ironmaking dictate that the iron ore feed material contain low amounts of gangue. Typically, good grades contain 65% iron and 5% gangue. Most Canadian ores contain from 23 to 40% Fe thus must be upgraded before processing.

During milling, large quantities of water are added (4). The amount varies with the method of upgrading, but a typical gravity-magnetic separation operation may use about 1.25 t of water per tonne of ore processed to the concentrate stage (5). Additional water is used during pelletizing; this amount can vary drastically depending upon whether the concentrate is reground for pelletizing and if the regrinding is wet or dry. When dealing with dry concentrate in pelletizing, water is added to maintain the moisture content from 7 to 9%. Much of the water can be reclaimed and recycled. Typically, about 75% of the water for the milling of hematite and up to 95% in some magnetite operations can be reclaimed (6).

Iron ore in Canada is mined from open pits except that produced as a byproduct from different operations. (The Iron Ore Recovery Plant built by Inco Ltd., at Copper Cliff in 1955, produces more than 2000 t of iron pellets per day.) The pit is divided into mineable blocks which are blasted and trucked to a crusher. The ore is then blended, crushed and ground to a size suitable for upgrading by gravity separation, magnetic separation, and flotation or a combination.

After crushing, the ore is usually ground to liberate the iron mineral from the gangue. Sizes may range from minus 2 mm for some gravity separation techniques to 44 µm for flotation. Magnetic cobbing is often done during the crushing stage at much larger sizes (4). The wet iron concentrate from the last stage of concentration is then filtered to remove water. The filter cake is usually unsuitable for ironmaking because its fine size would lead to high dust losses and plugging. Therefore the fine particles are agglomerated into larger sizes by two approaches. The fine particles are rolled into balls of about 8 mm diam which are hardened by heating to 1300C to enable them to withstand further handling, shipping and processing. Alternatively, the particles are mixed with fine coke, limestone and steel-plant wastes and heated to form fused lumps called sinter.

The iron blast furnace is charged with lump iron ore, pellets and/or sinter along with coke and limestone. An upward blast of preheated air partially burns the coke and generates temperatures in excess of 1500°C. Oxygen is removed from the iron ore by the partially combusted gases to produce molten elemental iron. The liquid iron forms a pool at the bottom of the furnace. The gangue combines with the limestone to form a liquid slag which floats on top of the liquid iron because of its much lower specific gravity. The iron (pig iron) and slag are tapped separately from the bottom of the furnace.

Liquid iron is then refined to steel by removing residual amounts of silicon and carbon with oxygen. Other deleterious elements such as phosphorus and sulphur must also be removed. Steel producton is normally carried out in the basic oxygen furnace (BOF), but the electric arc and open hearth furnaces may also be used. In the final stage, relatively small amounts of alloying elements, as ferroalloys, are added to provide the required properties of the various grades of steel. The molten steel is then cast into ingots or continuously cast slabs, blooms, or billets (steel shapes of different dimensions) (7).

Steel can also be produced in electric arc furnaces fed with steel scrap and/or sponge iron. Sponge iron is ore which has been reduced directly to metallic iron without melting using a variety of techniques (8). Direct reduction does not require coke however, none of the commercial processes can match the efficiency of a blast furnace. About eight different commercial direct reduction processes exist worldwide but in Canada only the Midrex process is used, at Contrecoeur, Quebec (9). Steel is produced by melting sponge iron and/or steel scrap in the electric arc furnace, and then refining the molten metal to remove deleterious elements using oxygen, adding alloys and casting as in BOF steelmaking.

At Stelco's Griffith Mine, a direct reduction kiln process known as the SL/RN was installed several years ago but has operated only occasionally when market demand and price were high.

Ferroalloys provide an inexpensive source of alloying elements in a form that is handled easily and dissolves readily in molten steel. The tonnage ferroalloys, i.e., silicon, manganese and chromium are made by smelting their oxide minerals with coke, limestone and iron ore in an electric arc furnace. The oxides are reduced to metal and combine with the iron to form a relatively low melting ferroalloy. Traditionally, high grade raw materials have been found necessary to meet the quality and price demands of the industry. Metals such as molybdenum, vanadium, tungsten and columbium (niobium) are produced by metallothermic reduction in unfired vessels using strong reducing agents such as aluminum and silicon. For all ferroalloys the molten product is allowed to solidify, broken into lumps smaller than 20 cm and sold in bulk or conveniently-sized packages.

PROBLEMS

HEALTH AND SAFETY

The iron ore industry enjoys a better than average safety record compared with other mining and heavy industries. This may be attributed to their aggresive safety campaigns and the fact that the mining is predominately open pit. Available figures indicate there are one third less accidents in open pit mines compared with shaft or underground mining (1). Recently, it has been realized that much of the gangue material is made up of amphiboles, e.g., anthophyllite and grunerite. These minerals are fibrous and are defined under asbestiform and could be a potential health hazard to the lungs of the miners (10). Further investigations are needed to determine the extent of this potential hazard.

The iron and steel sector does not have as good a safety record as the iron ore industry but it is considered good compared with other hot metal industries. The presence of molten metal and the use of very large equipment present special hazards.

ENVIRONMENT

The very size of the ferrous industry creates a problem in the handling of voluminous solid, liquid and gaseous wastes. Slags are relatively inert and can be used as structural materials and stable landfill. However, water from iron ore concentrators, especially hematite ores, contains very fine ore particles which are difficult to settle and can severely discolour local watersheds. In addition, vast volumes of cooling water are used in steelmaking, which must be cooled for reuse or discharged into the environment. Airborne dust from materials handling and process fumes from sinter plants, steelmaking and ferroalloy furnaces present problems of control, collection and disposal.

COMPETITION

The Canadian iron ore industry is in serious difficulty in trying to remain competitive with other world exporters, mainly Brazil which supplies approximately 20% of world export demand. Brazil has very abundant iron ore reserves including vast quantities of high grade ore which does not require concentration and pelletization prior to shipping (direct-shipping ore). The new Carajas Iron Ore Project in the Amazon basin will be coming on-line in the mid-1980's when a railroad is completed to the coast. With approximately 30 000 Mt of ore grading 66% Fe (11), Brazil may begin to seriously affect Canada's ability to compete because Canadian ores must be concentrated and pelletized before shipment at an extra cost in excess of $20/t of iron (energy, supplies and materials only) (12). Even companies that own Canadian iron ore mines are beginning to sign contracts for Brazilian ore.

Canada is reasonably competitive in ironmaking and steelmaking. Japan is a leader in technology because of its very modern facilities and the advances in Japanese R & D. Canada is competitive domestically and in the USA with Canadian steel supplying roughly 90% of domestic consumption and approximately 15% of all steel imports to the USA (2). Canada has been able to acquire new technology abroad and apply such technology effectively and efficiently. There is concern that Japanese technology is developing faster than Canada can assimilate it into existing facilities so that the technology gap is widening. A significant area is the production of very low-impurity steel.

SELF SUFFICIENCY

With reserves of more than 150 000 Mt (13) and a current consumption of 16 Mt/a (14), Canada has ample iron ore to meet its domestic needs for the foreseeable future. Current ironmaking and steelmaking capacities are more than adequate for industrial demands and could be readily expanded if the demand should increase.

For ferroalloys, Canada depends on imported chromium and manganese (15,16). The world's main sources of chromium are in politically sensitive areas such as the African continent and the USSR (15). Manganese has more varied sources but is also subject to the vagaries of international politics (16). Domestic sources of both chromium and manganese exist but the grades are low and process costs would make domestic products non-competitive (15,16). Canada is a major exporter of molybdenum and columbium (niobium) (3). Silica for ferrosilicon is abundant. Vanadium occurs in the Canadian oil sands bitumen but has not yet been exploited (17). In summary, existing technology has not overcome the economic disadvantage of Canadian production of chromium, manganese and vanadium.

HIGH-TECH IMPACT

The greatest effects on the milling industry would appear to be more sophisticated automated control of processes and the possible use of robotics where more hazardous or physically difficult tasks are required. The increased demand for lighter and stronger metals by large consumers such as aerospace and Department of Defence, e.g., ships, airplanes, ground transport, and weaponery would present an outstanding challenge to the industry to develop new processes and more varied alloys.

CONSERVATION

Recoveries of iron from ore are relatively low (about 70%) because of the separation processes (12). This means that a significant amount of iron is lost to mill tailings and is considered unrecoverable. Unpredictable and poorly defined variations in the ore as-mined can also cause iron losses in the concentration step.

Canada has abundant titaniferous magnetites which could become a future source of iron, titanium and byproduct elements such as vanadium, chromium and aluminum, but the current market does not warrant the expense of developing the new technology needed to exploit these deposits. Quebec Iron and Titanium currently produces iron and a valuable titanium-bearing slag from ilmenite (18,19).

Ironmaking and steelmaking slags are used locally as a replacement for structural fill, as pozzolannic cement, and in the manufacture of mineral wool. There is considerable scope for increasing their use since most of these slags are disposed of as onsite land fill.

The steel industry consumes the majority of steel scrap generated in Canada by remelting it. Remelting scrap requires considerably less energy than producing steel from primary iron ore. Such scrap also contains metals such as zinc, lead and cadmium which report to steelmaking dusts. Currently, these dusts present a problem for environmental reasons but they could be a source of metal values with the right technology.

ENERGY CONSERVATION

The largest energy-consuming step in the production of steel is the reduction stage. Theoretically the process should consume about 260 kg of carbon per tonne of iron. Prior to the development of metallurgical coke in the 18th century, some 6000 kg of charcoal was required per tonne of iron. In the 1950's, technology had reduced carbon consumption to 1000 kg/t and in the late 1970's Japan was reporting fuel rates of about 500 kg/t. Further improvements in technology are expected to produce only marginal changes in ironmaking energy consumption (20).

The induration of iron ore pellets is another major energy-consuming step, requiring about 1000 MJ of energy per tonne of pellets. Heat recuperation is being utilized more commonly now but heat losses still account for a large part of production costs (12).

ENERGY REPLACEMENT

To date there has been no effective means of replacing heavy fuel oil for pellet induration in the Quebec-Labrador iron ore industry. Alternatives such as plasma (electrically fired) have been evaluated by companies but no specific plans have been made.

In ironmaking, metallurgical coke has been partially replaced by fuel injection (fuel oil, natural gas and coal). Plasma-assisted injection is being proposed (9). The open hearth steelmaking process using fuel oil has been replaced by autogenous oxygen processes such as BOF. The Midrex direct reduction process uses natural gas as a reductant instead of coke, and the SL/RN kiln can use coal, oil, natural gas or a combination of fuels (9).

DIVERSIFICATION

The ferrous industry has not diversified significantly, the production of iron ore, iron and steel and ferroalloys remain the central business. Localized transportation systems to serve corporate needs such as railway haulage of iron ore have been developed. The Costell group, owners of Lake Ontario Steel, have diversified broadly into engineering, design and construction of steel plants. The Iron Ore Company of Canada has an aggresive program underway to look at silicon, ferrosilicon and strategic metals (12).

LABOUR

Iron ore operations are generally remote from major Canadian centres resulting in labour shortages. In an attempt to induce workers and their families, the iron ore companies used incentives such as tax-free transportation costs, subsidized housing and other benefits with the resulting maintenance of a relatively stable labour force. However, in 1980, with the enactment of the Northern Allowance Remission Order, all such benefits became taxable and combined with the higher cost of goods in these remote areas, the employee turnover now is approaching 40% (21). Although the order is under review, one company conservatively estimates by 1984, its employees will have lost $4 million in income (12). With such a high turnover, it is becoming increasingly expensive for companies to operate in remote areas, due to a continuous training program for new employees, with a resultant loss in productivity. The latest figures available report a total of 10 070 employees earning a total of $203 million in 1977 (18).

Conversely, the steel industry has a relatively more stable working force since it tends to be located close to markets and becomes the focal point for large industrial centres such as Hamilton. Labour rates for the relatively skilled worker in the steel industry are generally much higher than average. In 1977, iron and steel mills employed 41 275 employees earning a total of $678 million (18).

Although the industry has become more automated and the processes more remotely controlled, physically demanding jobs still exist in housekeeping and maintenance.

CURRENT RESEARCH AND DEVELOPMENT

Canadian research and development in the iron ore industry is minimal and consists mainly of process troubleshooting. Major or long term research is done by the US laboratories of the American owner companies, e.g., Pickands Mather & Co., Hanna Corporation, and Cleveland-Cliffs (18). In recent years, these laboratories have become preoccupied with non-Canadian problems and as a result the Canadian mining industry has become more interested in utilizing the expertise and facilities of CANMET. In 1979, an Iron Ore Technical Task Force was formed to discuss areas of mutual concern for the iron ore producers. Co-operative research projects have resulted in contributions from CANMET, Environment Canada, provincial agencies such as the Quebec Deptartment of Natural Resources, Centre de Recherche Minérale and the iron ore companies. For example, the red water study of Quebec-Labrador was a cooperative effort which involved seven mining companies, CANMET, Environmental Protection Service (Atlantic and Quebec regions) and Inland Waters (EC) of Burlington, Ontario. The total Canadian R & D complement devoted to iron ore processing is estimated at 10 PY (22). University research devoted to iron ore is undertaken by several universities across the country (23).

The Canadian iron and steel industry has about 250 researchers of whom an estimated 40% are working on process metallurgy. Stelco has the largest R & D complement with 125 persons at its Burlington, Ontario, research centre. CANMET has only a nominal number of people working in process metallurgy. Ferrous process metallurgy is a strong element of Canadian university metallurgical engineering courses and research programs (24).

GAPS IN TECHNOLOGY AND CHALLENGES

HEALTH AND SAFETY

Based on finding anthophyllite in the living tissue of a miner at Wabush (10), it would appear that both long and short term studies should be undertaken at the iron ore mines to see if this is a chronic hazard which has just now emerged or perhaps only an isolated case which may have been contracted off the mine site. Chest X-rays should be given on a regular basis to check against any hazardous materials.

Trends toward more automation and remote control should improve the health and safety record, e.g., development and use of robotics. Removing the worker from hazardous areas in the steel industry is a continuing trend.

ENVIRONMENT

The environmental problems in the ferrous industry are far from being solved. From the red water study mentioned previously, seven different chemical treatment strategies have been developed on a jar-test scale only, which would incur reagent costs on a per annum basis from $75 000 to $650 000 ($Can.) (6). These costs are based on an average flow of 750 L/s to achieve total iron less than 1 mg/L of effluent. The more expensive strategies incorporate a decrease in settling time which would reduce the size of tailings ponds with an overall capital savings. It must be stressed that these strategies have all been developed in a laboratory on a jar-test scale under rather quiescent conditions, but have not as yet been employed or even pilot tested under actual conditions. Cooperation and financial commitment by the industry is needed to do these pilot-scale studies.

At this point the Department of Environment has been able to ascertain that the only toxic effects of colloidal hematite in water appear to be on a large kill of salmon roe, however, studies are continuing to determine if further damage is being done to vegetation, and benthonic and planktonic life in the aquatic environment. Although human ingestion of this water so far appears to be harmless, long term effects are being studied to assure future generations do not suffer from chronic exposure.

Red fume from steelmaking has been reduced but levels could still be lower. Continuous processes are evolving which tend to minimize the number of sources of dust, and promote heat recovery.

There is a vital need for a more consultative and cooperative scientific effort in Canada to resolve inherent problems in the iron ore industry. Unilateral efforts and initiatives by government agencies will be insufficient to effect substantive change.

COMPETITION

The severest challenge to Canada's iron ore industry will come from Brazil. No new technology exists and it is difficult to visualize new processing techniques being developed within the next 20 years, which will enable us to compete with high grade (67% Fe and less than 1% SiO_2) ores from Brazil (11,18).

SELF SUFFICIENCY

The most recent appraisal of Canada's iron ore reserves and resources conducted in 1972 by the Geological Survey of Canada shows (13):

Iron Ore Resources in Canada (x 10^6 t)

	Measured		Indicated		Total	
	Crude	Fe	Crude	Fe	Crude	Fe
Status [1]	10 747	3 802	1 086	365	11 833	4 167
Status [2]	6 303	1 817	7 696	2 244	14 000	4 060
Additional potential ore [3]					128 761	37 119
	17 050	5 619	8 782	2 609	154 594	45 346

1 Exploitable under 1972 technologic and economic conditions.

2 High probability of becoming exploitable.

3 Additional resources that are potential future sources of supply.

Since 1972, there have been approximately 950 Mt of ore mined and an additional 2700 Mt is forecast to be extracted by 2000. Thus, Canada's resources are adequate for predicted domestic and export requirements into the 21st century. Reserve and resource definition is fluid with time and economics so that large quantities of iron ore currently defined as resources could change to reserves with newer and more sophisticated methods of technology (25). With the diversificaton of the iron ore industry it is expected that Canada will be able to reduce to a large degree or eventually completely eliminate the importation of specialized steels, alloys and ferroalloys (12).

HIGH-TECH IMPACT

Canada is becoming one of the more sophisticated nations in high technology. It is hopeful that this technology will play an important role in the recycling of steel scrap by developing methods to eliminate the deleterious material from the products. This is a highly competitive business and it must adapt rapidly to keep abreast of the USA and Japan in steelmaking.

CONSERVATION

Future improvements in iron recovery will require process changes. For example, as stated previously, recoveries of iron ore run from 70 to 75%. Using newer and more innovative technology up to 25% more recovery might be obtained. This improvement in recoveries will have to come from a superior technology in working existing and marginal deposits, and a more highly developed technology to recover liberated iron within the mill process which is presently reporting to the tailings, e.g., electrolytic methods of recovering colloidal hematite while solving the red water problem. This appears to be an area where a great deal of R & D is required. Iron reserves could be increased by 20% if new technology were developed for reworking old tailings. Direct use of waste is expected in the future, e.g., use of slag as a cement replacement, and development of technology to optimize the recovery of base metals from fume.

ENERGY CONSERVATION AND REPLACEMENT

Although widespread concern about energy shortages did not materialize until the mid-1970s, the Canadian steel industry has for many years been a leading practitioner of energy conservation and management. The efficient use of energy has always been a major concern, because energy accounts for more than 20% of the steelmaking costs. The annual fuel bill amounts to about $900 million. Canadian steelmakers are predicting less gains in energy efficiency by 1985 than are other industrial sectors. However, the steelmaking sector has no "easy" conservation measures left to introduce. The industry energy efficiencies are close to the technically achievable limits of existing production facilities and programs.

Approximately 85% of the raw steel in Canada is produced in the plants of five integrated steelmakers - Algoma, Dofasco, Sidbec-Dosco, Stelco, and Sysco. In 1979, speaking jointly through the Ferrous Industry Energy Research Association (FERA), these companies committed themselves to reducing energy consumption, per ton of raw steel produced, by 7.1% over the 1974 base, by 1985. These five companies had achieved a 4.7% reduction in energy consumption by the end of 1980.

The Canadian steel industry is highly energy efficient by global standards. Efforts to decrease further energy consumption are being hampered, however, by a shortage of capital brought about largely escalating steelmaking costs at a rate in excess of that of steel prices. Another factor, as seen by the steel producers, has been stringent government regulation, especially in the area of environmental control, which has also increased industry costs and infringed on the supply of available capital (26).

In past years, Canadian steelmakers have pioneered many developments that have resulted in improved energy efficiency. For instance, The industry was the first in the world to use prereduced pellets, to operate blast furnaces with low slag volume which reduced significantly both limestone additions and coke consumption rates, and to inject heavy oil into furnaces to bring down overall fuel rates. It also introduced to North America the concepts of self-fluxed sinter and a 100% pelletized blast furnace burden, both of which have had major energy conservation implications. Dofasco installed the first basic oxygen steelmaking furnace in North America. The domestic steel industry also pioneered the direct reduction process which employs very low gangue pellets to minimize energy loss in slag, and now operates three of these high energy efficient facilities.

More recently, Stelco invented and developed an entirely new concept in hot strip rolling known as the coilbox. This device has effected a major decrease in energy consumption as well as significantly reducing the capital cost of constructing hot strip mills. A number of other energy conservation innovations also took place within the Canadian industry.

The Canadian steel industry is currently participating in a joint research program with the Centre de Recherches Métallurgiques (CRM) at Liège, Belgium, on plasma injection of superheated gas into a blast furnace. Potential benefits of this plasma technology include:

- Reduced coke rate due to increased use of electric energy.
- Diminished coke oven emissions.
- Increased flexibility in fuel resource utilization.
- Reduced dependence on imported metallurgical coking coal from the USA.
- Extension of the life span of existing coking resources.

In BTU terms, the companies in FERA consumed an average of 20.51 million BTU's per net ton of raw steel produced in 1980, or a total of 302 trillion BTU's (equivalent to 52 million barrels of crude oil). Of this, 65% came from coal, 19% from natural gas, 10% from oil, and 6% from electric power.

The International Iron and Steel Institute statistical models indicate that 2.3 million BTU's per raw ton of steel can be saved by converting from ingot to continuous cast production.

Direct rolling of steel ingots represents a viable alternative to continuous casting and is already being practised by one of the FERA member companies. Direct rolling ingots to coils lowers energy consumption by reducing the number of reheat/rolling steps required to get from an ingot to finished product. Energy savings are estimated to be 1.2 million BTU's per finished ton.

Rapidly escalating costs of oil and natural gas will cause a shift to coal and electric power in future decades. A trend to increased electric arc furnace steelmaking technology is predicted, although this will be limited by the availability of scrap, sponge iron and electricity.

Higher utilization of coal (coke) in the steel industry will increase the availability of internally generated gas, reducing the consumption of purchased natural gas (26,27).

In the iron ore industry, it appears probable that with the increase in the price of fossil fuels, that the industry will become more dependent upon electrical energy, especially in the Quebec-Labrador and northern Ontario regions being located near the James Bay Project and additional sources coming from nuclear reactors (1).

DIVERSIFICATION

As mentioned earlier, it appears that the major iron companies are in the process of developing silicon, ferro-silicon and other strategic metals (12).

The steel industry will make an attempt to develop new products in their own areas of specialization, exemplified by the introduction of continuous coils of zinc-protected galvanized sheet in 1955 by Dofasco and Algoma's development of wide flange beams in 1961 (26).

LABOUR

The iron ore industry will continue to face problems in recruitment and retention unless the Northern Allowance Remission Order is rescinded (21).

It is projected that the Canadian steel industry will need to invest $15 000 million in 1980 dollars before the year 2000 if it is to remain in a competitive position to satisfy domestic and foreign markets. This will mean an increased labour force, more highly skilled and better trained to cope with new technological advances (26). Any R & D done at CANMET would have insignificant impact upon the labour situation.

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INDUSTRIAL MINERALS

GENERAL DESCRIPTION OF INDUSTRY

Canadian shipments of industrial minerals in 1980 were valued at $4200 million; non-metallic minerals (asbestos, barite, fluorspar, etc.) contributed $2530 million and construction materials (cement, lime, sand and gravel, stone) $1670 million. Shipments and dollar values are summarized in Table 6.

Table 6 - Canadian shipments of industrial minerals, 1980 (1)

Nonmetallics	tonnes	$
Asbestos	1 323 053	618 493 360
Barite	1 323 053	4 380 167
Fluorspar	-	-
Gypsum	7 336 218	39 538 688
Magnesitic dolomite	7 336 218	10 404 392
Nepheline syenite	599 699	15 935 661
Potash (K_2O)	7 201 217	1 020 705 248
Quartz	2 251 831	29 184 492
Salt	7 422 854	122 775 145
Sodium sulphate	480 666	30 468 339
Sulphur, elemental	7 655 723	444 095 310
Sulphur, smelter gas[1]	894 732	28 912 895
Sulphur, pyrites[2]	14 328	344 770
Other nonmetallics[3]	14 328	167 124 065
Total nonmetallics		2 532 362 532
Total construction materials		1 668 577 000
Total industrial minerals		4 200 939 532

1. Sulphur content. 2. Estimated sulphur content. 3. Includes titanium dioxide, talc and soapstone, peat moss and gemstones.

The 1980 production of industrial minerals, at $4200 million, represented 43% of the total value of minerals ($9700 million) exclusive of fuels, for a per capita value of $175.

Canada ranks second in the world production of a number of industrial minerals - asbestos (27.5%), potash (23.6%), elemental sulphur (16.8%), and gypsum (11.5%).

Industrial minerals contribute substantially to the Canadian economy. Principal production and trade statistics for 1979, the latest year for which complete statistics are available, are noted in Table 7.

The industrial minerals industry is composed of a number of large Canadian and multinational corporations employing hundreds of people, many intermediate-sized operations and, as well, some small operations with perhaps only 5 or 6 employees on a part-time basis. The 1979 Minerals Yearbook (2) notes there were 21 798 employees in the nonmetallic industry (exclusive of structural materials) working in 378 separate mines or plants. The total number of employees in the industrial minerals industry, which includes structural material operations, probably would be double the noted 21 798.

Industrial minerals are utilized in practically all industrial processes and products. General use categories and minerals are summarized in Table 8.

Table 7 - Statistical data, industrial minerals, 1979 (2)

	Industrial Minerals		Total minerals	%
	Nonmetallics ($000)	Structural minerals ($000)	($000)	of total
Production	1 354 921	1 312 061		
	2 666 982		18 399 832	14.5
Exports	2 170 800	NA	19 818 600	11
Imports, crude and fabricated	931 000	NA	12 446 500	7

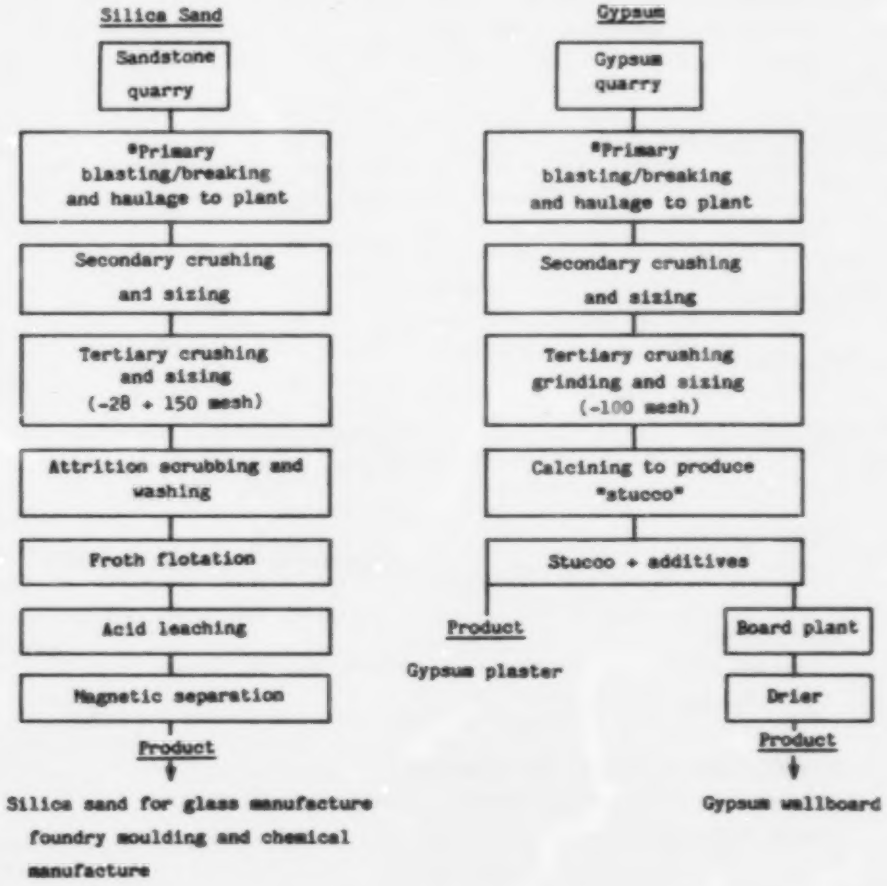
Table 8 - Uses/applications for some industrial minerals (3)

Application	Mineral
Abrasives	
loose grains, bonded and coated papers, and wheels	quartz sand, silicon carbide, aluminum oxide, garnet, topaz, talc, etc.
Asbestos/cement products	
brake lining, papers, pipe, cements	asbestos, silica
Ceramics	
clay brick, glass, sanitary ware, refractory products	clay, kaolin, silica sand, feldspar, nepheline syenite, etc.
Chemicals	
sodium hydroxide, chlorine, hydrofluoric, hydrochloric and sulphuric acids	salt, fluorspar, sulphur, etc.
Construction materials	
aggregate, concrete, gypsum and lime products, insulation materials, monuments, roofing granules	sand, gravel, crushed stone, slag, cement, gypsum, stone, etc.
Electronic and optical	
electronic equipment and applications	quartz
Fertilizers	nitrogen, phosphate, potassium, calcium, magnesium, sulphur, etc.
Fluxes	
metal production and forming	fluorspar, silica, lime/limestone
Foundry sand	
iron and steel and other metals, moulding, and fabricating	silica, zircon, olivine
Mineral pigments, fillers and extenders	
rubber, plastics, asphalt, paint, tile, gypsum products	most industrial minerals
Oil well drilling	
drilling aids and muds	barite, mica, asbestos, bentonite, diatomite, etc.

CURRENT TECHNOLOGY

Many industrial minerals, e.g., gypsum, are recovered from surface or near surface deposits by openpit mining. Potash is an example of others that occur only at depth, and traditionally are recovered by underground mining. In most instances, the bulk of the mined material is utilized, exceptions are asbestos and, to a lesser degree, potash.

Many industrial minerals are processed by dry methods including crushing and grinding, sizing by screens or air separators, and magnetic separation. Others may require wet processing such as jigging, attrition scrubbing, flotation and acid leaching. Some are processed further, e.g., by calcining to produce portland cement, lime, and gypsum plaster. Processing and recovery of a few industrial minerals are somewhat unique, e.g., sodium sulphate, recovered from brine lakes in Western Canada by concentration and evaporation; sodium chloride which, in addition to being mined, is recovered from underground brines or solution cavities and processed by multi-stage, vacuum pan evaporation; and elemental sulphur which is a coproduct of the production of natural gas from "sour" gas wells in Western Canada. Thus processing may differ markedly from one industrial mineral to another. Two typical flow sheets, one for silica sand, the other for gypsum/gypsum products are noted in Fig. 3.


Silica Sand
Sandstone quarry
*Primary blasting/breaking and haulage to plant
Secondary crushing and sizing
Tertiary crushing and sizing (-28 + 150 mesh)
Attrition scrubbing and washing
Froth flotation
Acid leaching
Magnetic separation
Product
Silica sand for glass manufacture foundry moulding and chemical manufacture
Gypsum
Gypsum quarry
*Primary blasting/breaking and haulage to plant
Secondary crushing and sizing
Tertiary crushing grinding and sizing (-100 mesh)
Calcining to produce "stucco"
Stucco + additives
Product
Gypsum plaster
Board plant
Drier
Product
Gypsum wallboard

* Processing may include washing to remove mud, salt, etc., and heavy media separation to remove anhydrite.

Fig. 3 - Typical steps in processing silica and gypsum

A few specific minerals are examined, with regard to processing problems, current R & D, gaps in technology, and challenges.

Because the industrial mineral sector is so varied, it is described under two major headings, each sub-divided into two components as follows:

Nonmetallics

- gypsum, lime, phosphate rock, potash, salt, silica
- asbestos*

Construction Materials

- aggregate, cement and concrete
- ceramics

*Asbestos, being one of the more important nonmetallic minerals, is considered separately.

NON-METALLIC MINERALS

PROBLEMS, CURRENT RESEARCH AND DEVELOPMENT, GAPS AND CHALLENGES

GYPSUM

Canadian gypsum production in 1981, from six provinces, was 7.3 Mt, valued at $39.5 million. Almost 70% is exported, chiefly from Nova Scotia, to the eastern USA. The remainder is used in Canada for gypsum plaster, wallboard manufacture, and in portland cement. Impurities typical of many gypsum deposits include anhydrite, limestone, mud and salt. These are reduced by selective mining, crushing and sizing and occasionally, by water washing on screens. Processing principally involves reduction to fine powder by crushing and grinding and, for gypsum products, low-temperature calcining (200°C). The calcined gypsum is slurried with water and formed into wallboard which is dried prior to shipment. Problems in the industry relate to quality of raw gypsum ore, high energy cost in processing and drying gypsum products, the development of new building products, and competition among producers. R & D during the last 8 to 10 years has centered around the development of continuous calcining systems to replace the batch, kettle system. Various types, including a "halo-flite" unit, one developed by National Gypsum and a third by Western Gypsum, are in use. Current Research and Development interest is in the field of energy conservation in calcining and, more critically, in drying gypsum board. An NEP CANMET project on conservation of energy in industrial minerals, ($1.5 million) probably will address this problem during the 4-year term of this program. Other areas of interest relate to the development of stronger, thinner wallboard and there appears to be some interest in sandwich construction, insulated gypsum building panels. Interest in phosphogypsum, a waste product from the phosphate fertilizer industry, is at a low level because of contained radium which discourages use of this material for gypsum products. A limited study of the feasibility of reducing radioactivity in phosphogypsum is planned by CANMET as part of a larger program on radioactive materials during FY 82/83.

LIME

Lime production in Canada in 1980 was 2.5 Mt valued at $129 million and was from 25 plants, principally in Ontario and Quebec, which together accounted for 85% of production. Lime is produced by calcining lump (crushed and sized) limestone in various types of kilns, the calcined product being used as flux for the steelmaking industry and in treating pulp for newsprint manufacture. Other uses include acidity control during processing for uranium ore, in domestic water and sewage treatment, and for the production of many calcium-based chemicals and fertilizers. A developing use is for SO_2 control in stack gas emissions from power plants. CANMET undoubtedly will become involved in this area of research in view of government's expressed concern re SO_2 pollution and acid rain. The development of uses for the resulting gypsum sludges also could be an area of study by CANMET.

PHOSPHATE ROCK (4)

The term "phosphate rock" is commonly applied to rock or granular material containing phosphate minerals. This material is mined and processed into a variety of products including elemental phosphorous, phosphoric acid, and phosphate fertilizers and chemicals. Canada has significant resources of phosphate rock but there is no domestic production; our entire requirements are imported, chiefly from the USA. Current requirements are in the order of 4 Mt/a, with consumption expected to increase to more than 5 Mt/a by 1990. Most commercial deposits are of the sedimentary type (carbonatites) and consist of loosely compacted, granular material. Processing is relatively simple and consists of water washing to remove extremely fine material, and flotation to recover a 30% plus P_2O_5 product. Domestic occurrences of interest are essentially restricted to Ontario, Quebec and British Columbia, with those of the Cargill area of northern Ontario being the most attractive for commercial development. These have been under intensive study during the past eight years by several Canadian companies. Although it has been shown that the production of a high-grade concentrate for the Canadian market is technically feasible, high capital and operating costs prohibit plant construction and production. Current studies relate to the possibility of producing byproduct vermiculite, clay minerals, quartz sand and columbium-bearing minerals that are associated with the orebody in significant amounts. In this regard, CANMET is engaged in a cost-recovery study to evaluate the technical feasibility of producing commercial-grade, expanded vermiculite from crude ore. Associated studies conceivably could be undertaken on the remaining minerals by CANMET during the next few years. The Cargill deposit lies on the Kapuskasing high, a several hundred kilometer geological structure extending from Lake Superior to James Bay. This structure is known to contain at least nine other carbonatite complexes. These deposits should be studied in detail to develop mineralogical and processing data which would substantially add to our knowledge and assist future development. A recently completed reference document suggests that by the year 2000, Canadian demand for phosphate rock may exceed the amount that Florida, in particular, is able to supply (4). The development of Canadian resources of phosphate rock is therefore imperative.

POTASH (KCl)

Canada's total current production of potash, about 7 Mt/a, is from underground deposits in Saskatchewan where there are 10 mines and one solution mining operation. In underground mining, continuous mining machines cut and load the potash which is raised to the surface, crushed scrubbed with a brine solution, and treated by flotation to separate the potash from the salt; the potash (KCl) is dried, sized and packaged for shipment to markets; the salt (NaCl) is sent to large surface storage areas. About 1.5 t of waste salt is produced for every 1.0 t of K_2O equivalent.

In solution mining, a hot solution is injected under pressure into a cavity in the potash beds through boreholes; concentrated solutions of salts are withdrawn from the bottom of the cavity and pumped to the surface for processing which consists of evaporation, crystallization, and salt separation; a portion of the sodium chloride is transported in slurry form to a neighbouring plant for the recovery of fine crystallized salt; the remaining salt brine is pumped to a large disposal pond.

Although processing of potash is relatively simple, concern has been expressed about the loss of potash values to the waste salt. In this regard, CANMET has initiated a three-year study of the surface chemistry of potash ore to determine whether improvements can be made in the flotation process. A second problem relates to the surface storage of vast quantities of waste salt. Wet processing underground is undesirable; however, if electrostatic separation or other type of dry processing techniques could be effected economically then the bulk of the processing could be carried out underground, the waste salt being left there as backfill in mined-out areas. Although dry processing has been studied in the past there is need for further study of this problem. Utilization of waste salt from the potash operations is also an area deserving of further attention. A small portion is used as de-icing salt but the bulk remains in open-pond storage. Although surface storage of waste salt/salt brine presents no serious problem at present, this salt nonetheless is a potential source of groundwater pollution and storage areas must be carefully controlled and monitored at all times. This problem could become critical during the next decade or two as potash production is expanded to aid and assist food production in third world countries.

SALT (NaCl)

Salt is recovered by underground mining at three operations, two in Ontario and one in Nova Scotia and, by brining, at six plants in

Ontario, Alberta, Saskatchewan and Nova Scotia. Production averages 7 Mt/a. Rock salt, a coarse salt for use in ice control on highways, is processed by crushing and sizing, the chief impurities, limestone and anhydrite, being largely concentrated in the coarser sizes which are discarded. High-purity "fine" salt for use in food processing is recovered from brine solution by multi-stage, vacuum pan evaporation. Impurities are eliminated in "purge" tanks prior to evaporation. Although problem areas are few, R & D in regard to dry separation methods to remove finer sizes of impurities from coarse crushed salt, and for further purification during wet processing, should be considered. Another area of interest is that of crystal growth during the evaporation/crystallization process. One company is currently investigating techniques for promoting larger size crystals which could, for example, be used to replace costly and energy-intensive "fused" salt, particularly in water softener applications. Fused salt is produced by melting evaporator salt in large oil- or gas-fired furnaces, cooling the molten salt by briquetting, and crushing and screening the resulting product to required size specifications.

SILICA

Silica is produced by only a few companies, principally in Ontario, Quebec and Manitoba, in lump form for use as metallurgical flux, and as crushed material or sand for the manufacture of glass, artificial abrasives, silica chemicals, etc. Production in 1980 exceeded 2 Mt which was less than 50% of the total Canadian consumption. The remaining requirements, imported from the USA, mostly consisted of high-quality sand for glass manufacture and for use by the foundry industry.

Production of lump material is relatively simple, the main requirement being that of accurate sizing to meet consumer specification, e.g., -10+5 cm (-4+2 in.) for ferrosilicon manufacture. Production of silica sand is more complex and requires size reduction of source material, which may be quartzite, quartz, sandstone or sand, to a specific size requirement, e.g., -700+104 µm (-28, +150 mesh) for glass manufacture, followed by scrubbing, flotation, magnetic separation and, on occasion, acid leaching to remove impurities to meet rigid customer specification, e.g., 0.02% Fe_2O_3 for glass manufacture.

Canada does not have ideal resources of silica material to meet the need of industry - which requires 50% importation of high-purity silica sand. Domestic resources suitable for use, e.g., offshore marine deposits in the vicinity of the Magdalen Islands in the Gulf of St. Lawrence and the Potsdam sandstones of eastern Ontario and southern Quebec, are extensive but require fairly costly treatment to render them acceptable for the principal markets - glass manufacture and foundry moulding. High silica content (99.6% SiO_2) and low impurity content (particularly iron and alumina) are mandatory for the glass industry; high silica content and proper grain shape and size distribution are required for steel foundry applications. Iron and alumina (feldspar) impurities in the Magdalen sands must be removed by flotation to render this material suitable for use as foundry sand. A further reduction of the iron, by acid leaching, may be required to produce glass-grade silica sand. Pyrite (iron sulphide) and alumina are typical impurities in Potsdam sandstone. These must be removed or reduced, often with considerable difficulty. The Potsdam sandstone, reduced to sand size, is not ideal for foundry sand applications because of size distribution and grain shape, however, it is being used more extensively today, primarily because of the increasing high cost of imported sand.

Resolution of this problem requires an intensive study of Canadian silica resources to determine whether improved processing can assure production of both glass and foundry sand from these resources. CANMET recently completed a cost-recovery project of the processing and beneficiation of Magdalen Islands sand for foundry and glass sand markets and is currently investigating other selected deposits of silica as part of a 3-year program. This program is being carried out in cooperation with provincial governments. Processing and beneficiation involves reduction of material from the silica resources to sand followed by attrition scrubbing, desliming, flotation and magnetic separation. In certain instances, acid leaching may be required to reduce iron to 0.02% Fe_2O_3, the target specification for many uses. Improvements are possible in two areas in particular, flotation and acid treatment. Both are or will be studied intensively during the course of this investigation. Of developing interest is the production of extremely high purity silica for use in the manufacturing of fibre optic filaments, and silicon for solar cell applications. CANMET recently initiated a limited study in this area which will involve chemical treatment of higher quality silica materials. This work is considered to be important and should be enlarged in 1983 and subsequent years as interest increases and as more resources become available.

INDUSTRIAL MINERAL RESOURCE BASE: GENERAL R & D REQUIREMENTS/NEEDS

Canada continues to import significant quantities of some minerals which, although available from domestic deposits, are not exploited to the degree that they could be, e.g.. phosphate, fluorspar, kaolin, silica. An extensive R & D program had been initiated on domestic silica resources. This should be followed by equally extensive studies on resources of other industrial minerals in Canada. The objective of such studies would be: 1) to develop basic background on Canada's resources of industrial minerals and the technology required to utilize these resources; 2) to identify existing or perceived problems re supply sources, beneficiation requirements, market opportunities, etc., and 3) to undertake the R & D required to ensure optimum utilization of these minerals in the domestic and foreign market place. In this regard, CANMET is identifying the R & D needs of the industrial minerals industry. Returns from a questionnaire to the industry are being received and compiled and a report will be available in early 1983. Returns to date note R & D needs in a number of areas and provide some information re actual R & D by companies. This information is briefly tabulated in Tables 9 and 10.

Table 9 - R & D needs of the industrial minerals industry

Mineral	R & D needs identified
Asbestos	- removal of mica impurity from fibre
	- efficient and economic separation of fibre at 74 μm (-74 μm, -200 mesh dust considered harmful)
	- development of new asbestos-based products
	- development of chemically modified fibres
	- use of asbestos fines (5 μm fibres) in plastics
	- screening of damp material (2 to 3% water) on 589 and 417 μm (28 and 35 mesh) sieves.
	- drying of fibre to reduce degradation with regard to efficiency and economics
	- improved filtration of asbestos-cement slurries (products) - use of asbestos wastes in soil conditioning/fertilizing
	- energy conservation
Barite	- development and beneficiation of deposits in Atlantic Canada for use in oil-well drilling

Table 9 (Cont'd)

Mineral	R & D needs identified
Bentonite	- physical characteristics vs acceptability as oil-well drilling mud - chemical reactions between nuclear wastes and bentonite - use of bentonite as sealant in landfills, lagoons, ponds for waste and tailing storage - greater utilization of Western bentonites in Canada - use of bentonite as forest fire retardent
Gypsum	- quality of ore reserves in some areas - blasting/crushing/processing modification to reduce fines production - pelletizing/crushing of fines for recycling - markets for fines-soil conditioning/reclamation - removal of impurities - silica, silt, mud, salt - process monitoring controls, energy cost reduction, gypsumboard paper cost reduction, reduction of board weight - use of byproduct, TiO_2 and SO_2 abatement gypsum
Cement	- low purity of limestone in some areas - high sodium/potassium oxide and chloride - expansion of use of portland cement in construction - reduced consumption of fuel and electrical power - limited supplies of gypsum for use as retarder in Western Canada - fly ash and other additives to reduce consumption of portland cement in concrete - use of fine limestone additive to replace up to 5% of portland cement in concrete - can this be increased to a maximum - utilization of high-alkali waste kiln dust, possibly as soil conditioner/fertilizer and as feed supplement
Lime/limestone	- handling of dust/sludge from collecting systems at lime plants - utilization of same - conservation of energy in lime plants - expanded use of lime to combat effects of acid rain - limestone, control of oversize during breaking - control of sulphur and magnesium contents for product quality and emission standards - development of alternative fuels - municipal solid waste, pulverised coal, petroleum/coke mixtures

Table 9 (Cont'd)

Mineral	R & D needs identified
	- information re blending of limestone/fertilizer to meet specific needs
	- use of limestone in SO_2 abatement systems
Nepheline syenite	- ultra-fine classification for production of minus 10 micron product
	- use as filler in plastics
	- reduction of iron minerals in hornblende nepheline syenite - use of byproduct biotite
	- utilization of tailings
Silica	- beneficiation and agglomeration of fines, especially at plants providing lump material for ferrosilicon/silicon manufacture
	- location/development of high grade quartz deposits suitable for silicon metal manufacture
	- production of ultra high purity silicon for silicon chip manufacture
	- beneficiation of specific silica resources to meet market specifications
	- beneficiation/utilization of silica in fibre optics and in solar cells
Sodium chloride	- control of roof rock in underground mining
	- reduction of diesel emission contaminants and reduction of NO_2 from blasting
	- consolidation/compaction of waste fines
	- reduction of energy cost in brine evaporation
Sodium sulphate	- occasional high magnesium in brines
	- soil entrapment of salt crystals in ponds
	- filtration of brine concentrates
	- energy costs
Talc	- grinding/beneficiation to produce higher quality products

Table 10 - R & D studies by industry

Mineral	
Asbestos	- use of fine fibre as reinforcement in plastics - asbestos tailings for mineral wool manufacture - recovery of magnesium from tailings - recovery of magnesium compounds (MgO, MgOH, and $MgCO_3$) from tailings - production of refractory products from tailings
Bentonite	- use of bentonite in the storage of nuclear wastes - use of bentonite as a carrier for sodium selenate (fertilizer additive for sheep-grazing land) in Eastern Canada
Gypsum	- recycling of waste products (board) and development of required processing equipment
Cement	- addition of carbonate to normal portland cement to meet revised CSA Standards
Lime/limestone	- conservation of energy in lime plants, cheaper fuels, kiln characteristics - improved flame control in rotary kiln to minimize fuel consumption and to maximize emission control and product quality - use of heat reclaimers - blending of soil nutrients to meet present/future agricultural needs - beneficiation of crushed limestone to reduce impurity content
Shale/lightweight	- pelletized kiln feed
Aggregate	- density reduction of aggregate - conservation of energy
Sodium chloride	- crystal growth studies relative to crystal size in brine evaporation plants - larger crystals desirable
Sodium sulphate	- heat recovery and reuse
Talc	- wet processing (grinding and flotation) to improve product quality - removal of dark impurities by dry beneficiation to improve product - development of more finely ground material for specialty applications, particularly as mineral filler/extender in paints, plastics, etc.

All respondents reported common problems relating to competition, increased cost of transportation and energy, the need to develop electronic process control equipment to improve product quality and performance, and the increasing burden of higher taxes.

MINERAL FILLERS/EXTENDERS/ADDITIVES

Developing interest by industry in the use of additive materials to improve performance and decrease consumption of more costly components in various systems (e.g., mica and graphite fibres as fillers in plastics and rubber, and certain mineral waste products as fillers in asphalt mixes, to decrease consumption of resin, latex and bitumen, and bentonite and other additives to aid pelletizing of iron ore and fertilizer prills) has shown a need to more precisely characterize these materials, to understand their precise function, and to improve that function where possible. Surface characteristics, particle size and shape, and size distribution all play important roles. Research in this area by CANMET is mandatory to ensure a more precise understanding of the interaction of these materials in specific systems and to aid industry in the development and fuller utilization of mineral filler minerals.

MINERAL WASTE RESOURCES

Canada's diminishing resources of favourably located ore-grade reserves of many minerals, because of depletion, urban encroachment, etc., is attracting attention to the possible economic use and/or recovery of minerals and mineral material from lower grade deposits and, as well, from our substantial reserves of mineral waste, e.g., waste rock and mill tailings, slags, flue dusts and chemical sludges. Mineral wastes are being produced at a rate in excess of 700 Mt annually with less than one-half of 1% being utilized. These mined and semi-processed mineral materials have been documented in CANMET's Mineral Waste Resources of Canada report series (4-12). CANMET should now embark on a long-term R & D project to examine select wastes in detail and to develop the technology necessary to ensure the reuse of such wastes and thus to effect conservation of our nonrenewable energy and mineral resources. Research in this area is considered to be of high priority in many of the more developed nations of the world.

ASBESTOS

GENERAL DESCRIPTION

Asbestos is a commercial term applied to the asbestiform or fibrous varieties of several specific silicate minerals (13). The principal variety is chrysotile from the serpentine group, a hydrous magnesium silicate. Other commercial varieties (all amphiboles) are amosite, a trade name for a complex iron magnesium silicate (mostly cummingtonite-grunerite), and crocidolite, a sodium-iron hydrous silicate variety of riebeckite. Of minor importance are tremolite and anthophyllite asbestos.

Chrysotile, the principal variety, is graded and grouped according to fiber length. Most of the groups are divided into subgroups to comprise the commercial specifications. The particular properties which give asbestos commercial value are fibrous structure, high tensile strength that imparts reinforcing characteristics to numerous products, and resistance to high temperature and to certain types of chemical attack (14).

Long fibres (10 to 20 mm) Crudes Nos. 1 and 2 and Group 3 are used in the textile industry, as electrical insulation, filtration medium and reinforcing fillers in asbestos cement products where great strength is required.

Medium-length fibres (3 to 10 mm), Groups 4, 5, 6 are used as reinforcing fillers in asbestos-cement products, in friction materials such as brake linings and clutch facings and in paper and pipe coverings.

Short fibres (less than 3 mm), Groups 7, 8, 9 are used as reinforcing fillers in plastics, floor tile, asphalt, and in paints and oil-well muds.

Asbestos has more than 2000 uses, with the construction and building industry sector being the main market. About 80% of asbestos production goes to this sector for use in asbestos-cement products, asbestos papers and felts, floor tiles and other asbestos products. About 12% is used for friction materials and about 3% each for textiles and plastics with minor uses absorbing the remainder.

The total world production in 1980 was estimated at 5 Mt, of which chrysotile made up about 90%. Chrysotile is the only variety mined in Canada. The remaining production consists of about 6% crocidolite (blue asbestos) and 3% amosite with the other types amounting to less than 1%. Canada accounted for about 30% of world production in 1980 with the USSR, the world's largest producer, accounting for about 50%. Canada produced about 1.3 Mt in 1980, of which about 95% was exported. Production was valued at about $641 million and was the eighth most important non-fuel mineral produced in Canada. In 1980 the asbestos mining and milling sector employed about 8000 persons. In Quebec, which produces about 90% of Canada's asbestos generally from open pits, about 30% of the rock mined is milled, exclusive of over-burden, giving an average yield of about 6% fibre (15,16).

CURRENT TECHNOLOGY

Chrysotile is an asbestiform variety of serpentine and is normally associated with massive serpentine. It can occur either as "cross fibre" or "slip fibre". In the former occurrence, which includes most deposits of economic significance, the asbestos is distributed through the gangue matrix in veins varying in width from 0.2 mm to more than 25 mm. The recovery process is, therefore, unique in that it usually involves separation of a fibrous mineral from a massive form of the same mineral. Because the fibre has the same density and chemical composition as the host rock, advantage is taken of the physical difference of the rock and fibre after the latter has been released under impact and fluffed up in a dry state. In this state, the asbestos is lighter in bulk density and amenable to separation from the serpentine rock by screening and air suction. The essential stages of present day dry milling operations consist of crushing, screening, separation, fiberizing, grading and packaging (17,18).

Wet processing of asbestos ore has also been used on occasion for the recovery of fibres. Although wet processing has not yet found acceptance for large-scale commercial application in Canada, research work was carried out in the mid 1920s on wet processing run of mine ore (19). Wet processing was used for a short time by Johnson's Company Ltd. in the early 1950's and by Asbestos Corporation Ltd. in the 1960's for treating mill tailings on a small scale to recover short, commercial-grade fibres. A wet process mill has been in operation in the USA since the late 1960s using a beneficiation technique developed by Union Carbide Corporation to treat the short-fibre chrysotile of the Coalinga district of California (20). The short fibres are shipped as high density pellets which are used as a reinforcement agent in thermoplastics. Currently a wet process is under development by an Australian asbestos producer to produce a full range of fibre grades from run of mine ore.

Much of the development of improved and new process equipment, technical knowledge and expertise has taken place in Canada. Advances in Canadian technology over the past 10 to 15 years and their introduction into asbestos mills has included the development of equipment using fluid bed techniques for drying ore and separating fibre from the gangue (21); development of preconcentration techniques to lower cut-off grades in mining to improve overall recovery (22); improvements to grading and dedusting equipment; development of better control systems and quality control (23); improvements in packaging and shipping techniques; and improvements in the overall dust collecting systems to bring down fibre levels in the air, in-plant and in the general environment within those specified in the appropriate regulations. Fibre counts have successively been lowered from 10 fibres/cc to 5 fibres/cc and most recently to 2 fibres/cc and may be further lowered to 1 fibre/cc or even 0.5 fibre/cc.

It can be said that there is no standard method of milling asbestos. Each operation is designed to suit the rock from an individual deposit and must be sufficiently flexible to meet varying conditions. Consequently, detailed flowsheets, and selection and use of equipment components vary throughout the industry. However, the basic operations of crushing, drying, fibre separation, cleaning, fiberizing, grading and packaging are common to all mills.

A dry asbestos milling process, by its nature of handling dry and fine material, can be an extremely dusty operation. The dust collection equipment must be designed to confine the dust at its source, to maintain adequate suction to capture the dust, and to effectively remove the dust once it is captured.

The dust-laden air from all of the various processing and dust control sections of the mill is pulled through an air filtering system. There are usually 10 complete air changes in the mill every hour. The filtered air can be either discharged to atmosphere during warm weather or recirculated to the mill as heating air during cold weather with consequent energy savings. Current in-plant regulations restrict average concentration of asbestos fibre in the air to 2 fibres/cc, a condition met by current technology (24).

The processed fibre products can be classified into five major groups according to fibre length distribution. These are spinning fibre (Group 3), asbestos-cement (Group 4), paper fibre (Group 5), paper and shingle fibre (Group 6), and shorts (Group 7). These major groups are further broken down into about 30 grades according to fibre length distribution. These grades are divided according to other important fibre characteristics, such as degree of fiberization or openness, cleanliness and strength, to meet the special requirements of the consumer. Tests to measure these fibre characteristics include: wet classificaton, fines content, bulk density, air permeability, filterability, absorption, viscosity, grit and spicule content, colour, magnetic rating and strength in asbestos-cement products (25).

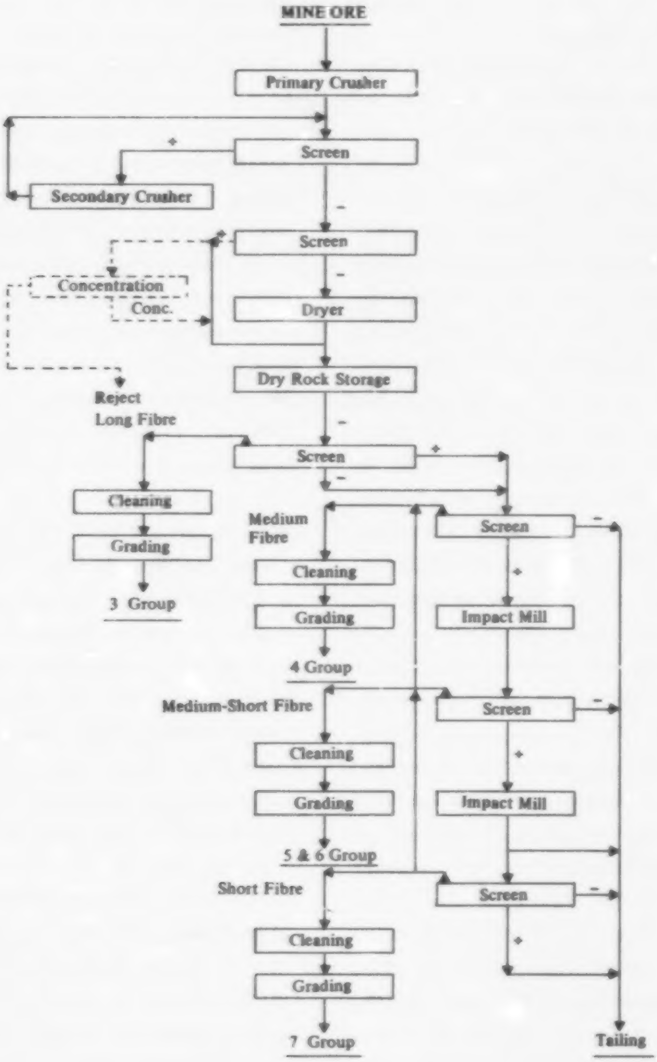

MINE ORE
Primary Crusher
Screen
+
Secondary Crusher
-
Screen
+
Concentration
Conc.
-
Dryer
Dry Rock Storage
Reject
Long Fibre
-
Screen
+
-
Cleaning
Grading
3 Group
Medium
Fibre
Screen
-
+
Cleaning
Grading
4 Group
Impact Mill
Medium-Short Fibre
Screen
-
+
Cleaning
Grading
5 & 6 Group
Impact Mill
Short Fibre
Screen
-
+
Cleaning
Grading
7 Group
Tailing

Fig. 4 - Simplified flowsheet, Canadian asbestos mill

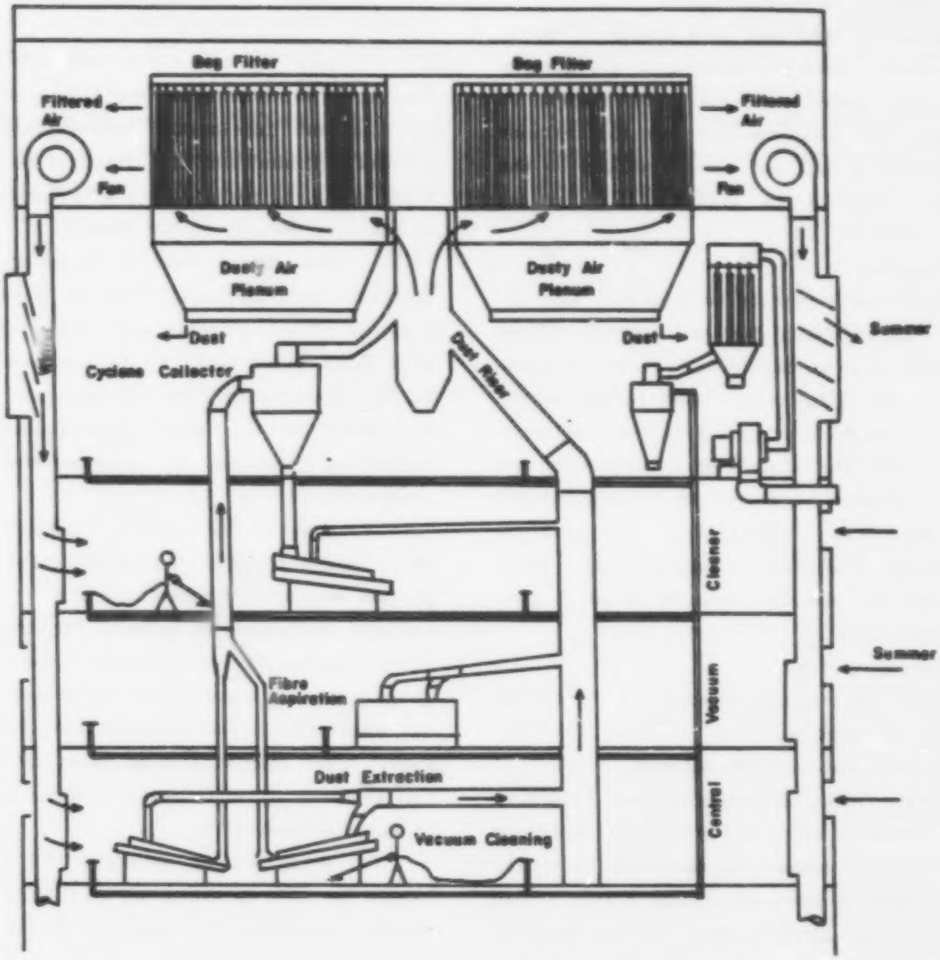

Bag Filter
Bag Filter
Filtered Air
Filtered Air
Fan
Fan
Dusty Air Plenum
Dusty Air Plenum
Dust
Dust
Cyclone Collector
Dust Riser
Winter
Summer
Summer
Cleaner
Vacuum
Control
Fibre Aspiration
Dust Extraction
Vacuum Cleaning

Fig. 5 - Schematic of air-handling and central vacuum cleaning system

PROBLEMS

1. **Working conditions:**
   - **Improved monitoring and control of fibre levels in the workplace**
   - **Improved dust collection systems to meet the more stringent regulations being considered**
2. **Environment:**
   - **Reduce stack fibre emissions to the atmosphere**
   - **Reduce wind borne dust from tailing piles**
3. **Maintaining competitiveness:**
   - **Automation to achieve productivity improvements (26)**
   - **Development of appropriate measuring devices to allow for automatic control systems to improve product uniformity, homogeneity and quality**
4. **Self sufficiency:**
   - **Canada has ample reserves of about 1500 Mt of ore containing about 50 Mt of recoverable fibre**

5. Exotic mineral-based materials:
   - Asbestos has been used in a number of exotic materials. However, because of the adverse publicity given asbestos with regard to health, new products and new applications for asbestos have not been developed to the extent possible in light of its unique properties.
6. Conservation:
   - Recovery could readily be improved but markets, particularly for the shorter fibres would have to be developed. Nickel is present in the ores as a nickeliferous magnetite, a sulphide or as a silicate. Attempts to recover the nickeliferous magnetite and the sulphide have proved to be uneconomical to date (27). Utilization of tailings including their use in mineral wool manufacture (28) and for MgO and Mg metal production (29).
7. Energy Conservation:
   - Improved drying and comminution to reduce energy consumption.
8. Sale of Machinery and Technology:
   - Canadian technical knowledge and expertise has resulted over the years in contracts for Canadian consulting engineers to design and construct asbestos milling plants. This in turn has led to orders for the specialized milling equipment usually available only from Canadian machinery manufacturing companies who have the manufacturing rights. However, subsequent orders have not always been forthcoming as the country concerned has developed its own manufacturing capability to produce the specialized equipment which in most cases is comparatively easy to fabricate once the design is available.
9. Working in Remote Areas:
   - At the Asbestos Hill mine operation, Ungava Bay, mining and a concentration stage take place over a nine month period. The concentrate is shipped to Germany for further processing to the various finished grades for the European and Middle East Markets (30).

CURRENT RESEARCH AND DEVELOPMENT

Very little R & D is underway in the processing of asbestos ore. Individual companies are carrying out incremental improvements to their process equipment and mill circuits to improve the fibre produced, particularly with further lowering of the dust content of the fibre.

The Trans Pacific Asbestos Inc. of Australia has completed research work on a wet process. They currently are testing the process on a pilot plant scale to demonstrate its feasibility in producing fibre suitable for the asbestos/cement market. Small tonnage lots are being tested and evaluated by asbestos/cement manufacturers (31).

Most of the current asbestos R & D projects relate primarily to health problems. These include medical and biological effects of asbestos as well as improved methods for measuring the fibre concentration levels in air and water. Projects are also underway to develop new products and applications for asbestos and asbestos tailings. About $6.5 million is being spent on asbestos research of which about 90% is spent in the Province of Quebec (32).

Breakdown of asbestos R & D effort in Canada for 1981 is estimated as follows:

Federal government (Intramural, grants & contracts)	$ 500K
Province of Quebec (SMA, IRDA, QAMA)	$ 4800K
Province of Ontario (Royal Commission on Asbestos)	$ 300K
Industry (in plant R & D)	$ 500K
Provincial research institutes and universities	$ 400K
TOTAL	$ 6500K

GAPS AND CHALLENGES

Three major challenges must be met by the asbestos mining industry:

1. Produce fibres of superior quality, particularly with lower dust content and improved filterability of A/C grades at a competitive cost, while meeting appropriate regulations

for airborne fibre concentrations in the workplace and in the general environment which in all probability will be lowered from the current 2 fibres per cc to 1 fibre per cc level or even further to 0.1 fibre per cc.

2. Develop new and improved products and applications for asbestos fibre and the utilization of asbestos tailings to off-set the possible phasing-out of existing asbestos containing products due to the possible health risks associated with asbestos.
3. Counter the many controversies concerning asbestos in relation to health, safety and the environment.

AGGREGATE, CEMENT AND CONCRETE

GENERAL DESCRIPTION OF INDUSTRY

The concrete industry in Canada has existed since 1889 when the first portland cement plant was established in Hull, Quebec.

At present, there are 24 cement plants with 51 kilns located in all provinces except Prince Edward Island. The production of cement in 1981 was 10.37 Mt valued at $680.6 million. In the 1970s many technological advances were made by the Canadian cement industry and new equipment installed. Continuing export of cement to the USA was advantageous.

It may be worth noting the degree of foreign ownership of the Canadian cement and concrete industry: more than one half of Canada Cement Lafarge and of Genstar, and almost one half of St. Lawrence Cement.

Aggregate is produced by about 300 firms in quantities ranging from 20 000 to 4 Mt/a. All districts are served by local pits and quarries with delivery distance rarely in excess of 50 km (33).

There is little lightweight aggregate manufacture in Canada; in recent years, some plants have closed. In 1981, there were 7 plants in operation with the production of expanded clay, shale and slag totalling 521 000 m^3 valued at $8.5 million.

It is relevant to note that ready-mix concrete suppliers are generally small firms: 40 to 60 trucks in metropolitan areas and possibly only 6 in rural districts (33).

Many block plants are also small and sell to a local market. Precast concrete plants producing structural elements are larger and typically have 50 to 100 employees but some are larger still and ship their products over a distance of several hundred kilometres (33).

There is one special feature of the Canadian cement, aggregate and concrete industry: the so-called vertical integration, i.e., the five largest cement companies own or control an undisclosed number of ready-mix, concrete and aggregate business establishments with perhaps 60% of the value of output (33).

Since the cement, aggregate and concrete industry is a supply industry to the construction sector, the cement, aggregate and concrete activity is very largely dependent on the overall construction activity. While this is also true of structural steel, the steel industry, unlike the cement industry, has many non-construction outlets.

Canadian shipments of construction materials are shown in Table 11. Exports of cement and clinker to the USA have played a great part in the successful development and expansion of the Canadian portland cement industry. Record amounts were exported during 1979, principally to New York, Michigan, Vermont, Washington, and the Canadian industry has been lamenting the fact that these amounts have not been duplicated since that time because of the recessionary conditions in the USA (Table 12). Exports of cement were expected to be reduced by about 7% in 1981, while clinker exports, principally to Canadian owned grinding subsidiaries in the USA, will likely be down more than 25%. If so, these figures will be 40% and 66% lower, respectively, than the 1979 records.*

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* Source: Mineral Policy Sector, Energy, Mines and Resources Canada.

Table 11 - Canada, shipments of construction materials, 1980-81

	1980		1981*	
	tonnes	$	tonnes	$
Cement	10 274 000	581 372 000	10 368 000	680 587 000
Sand and gravel	276 452 000	508 364 000	287 468 000	547 096 000
Stone	103 366 000	341 156 000	94 579 000	345 518 000
TOTAL		1 430 892 000		1 573 201 000

* Preliminary

Source: Mineral Policy Sector, Energy, Mines and Resources Canada.

Table 12 - Canada, cement shipments, 1980-81

	1980		1981*	
	tonnes	$	tonnes	$
Ontario	3 397 000	163 056 000	3 620 000	200 115 000
Alberta	1 155 000	97 000 000	1 400 000	135 409 000
Quebec	2 765 000	128 999 000	2 527 000	128 973 000
British Columbia	1 351 000	90 881 000	970 000	81 546 000
Manitoba	510 000	34 680 000	650 000	50 904 000
Saskatchewan	370 000	30 200 000	398 000	37 413 000
Nova Scotia	370 000	18 379 000	398 000	26 323 000
New Brunswick	370 000	11 762 000	398 000	12 516 000
Newfoundland	370 000	6 415 000	398 000	7 388 000
TOTAL	10 274 000	581 372 000	10 368 000	680 587 000

* Preliminary

Source: Mineral Policy Sector, Energy, Mines and Resources Canada.

CURRENT TECHNOLOGY

Portland cement is produced by calcining an accurately proportioned, finely ground mixture of limestone, silica, alumina and iron oxide in a rotary kiln to produce clinker, a fused chemically complex mixture of calcium silicates and aluminates which, in turn, is finely interground with gypsum to form cement. This material is then mixed with sand, aggregate and water to form concrete. A schematic outline of cement production is shown in Fig. 6 and a schematic outline of conditions and reactions in a typical rotary kiln (dry process) is shown in Fig. 7.

PROBLEMS

The problems associated with the processing of cement and lightweight aggregates relate mainly to the very high energy costs (34). To produce 1 t of cement requires 4×10^9 J (or 4 GJ) of energy. Also, environmental control requirements to minimize the amount of alkali in the flue dust have increased the alkali content of cements and resulted in potential alkali-aggregate reaction problems. During the past 10 years, all light-weight aggregate processing plants have closed in Ontario and Quebec due to high energy costs and local clean air bylaws. In the manu-

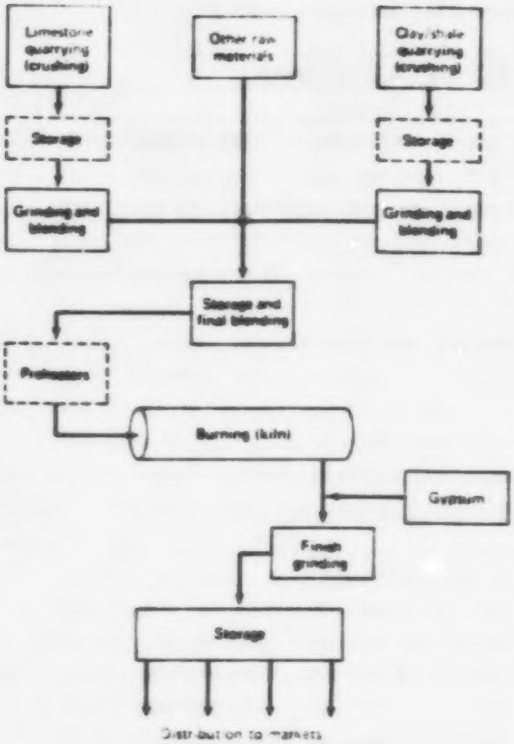
Limestone quarrying (crushing)
Other raw materials
Clay/shale quarrying (crushing)
Storage
Storage
Grinding and blending
Grinding and blending
Storage and final blending
Preheaters
Burning (kiln)
Gypsum
Finish grinding
Storage
Distribution to markets

Fig. 6 - Schematic outline of cement production (from Concrete by Mindess & Young, 1981)

facture of normal weight aggregates, there is the unresolved issue of what to do with waste limestone dust. Several million tonnes of this dust is stockpiled in limestone quarries. Further needs include the development of high-strength, durable concrete for construction in the North and for offshore drilling platforms, and a study of the problems of finding suitable aggregate sources for concrete in northern Canada.

CURRENT RESEARCH AND DEVELOPMENT

The total expenditure for R & D in cement, aggregate and concrete in Canada is in the order of \$3.5 million annually. This should be viewed against the value of cement production, \$650 million (estimated), and that of construction, \$5000 million (1980). The breakdown of the research effort for 1980 is as follows (33):

Industry	- \$1 296 000*
Federal agencies	- \$ 651 000
Provincial agencies	- \$ 947 000
Universities	- \$ 577 000
	\$3 471 000

(*This includes research done by a large number of small and large companies across Canada and covers all aspects of the concrete industry.)

Most research by provincial agencies and industry is concerned with solving specific problems and, as such, has no Canada-wide application.

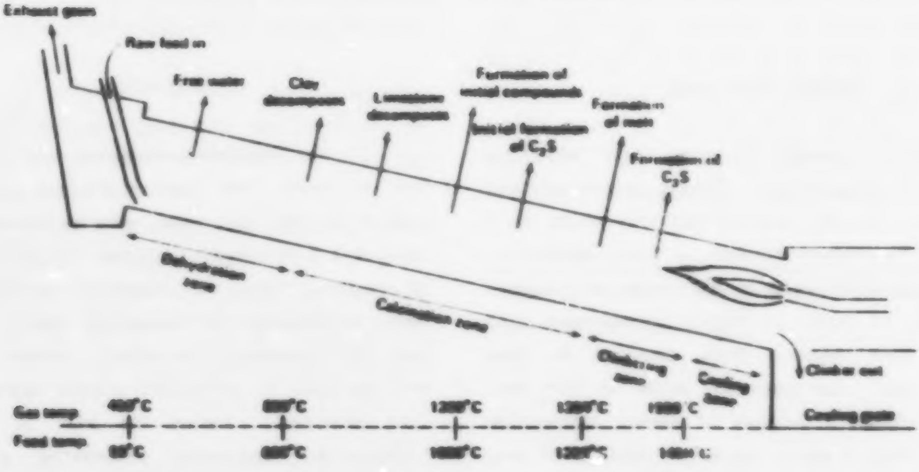
Exhaust gases
Raw feed in
Free water
Clay decomposes
Limestone decomposes
Formation of initial compounds
Initial formation of C_2S
Formation of melt
Formation of C_3S
Dehydration zone
Calcination zone
Clinkering zone
Cooling zone
Clinker out
Cooling grate
Gas temp
Feed temp
450°C
800°C
1200°C
1800°C
800°C
1000°C
1200°C

Fig. 7 - Schematic outline of conditions and reactions in a typical cement rotary kiln (dry process) (from Concrete by Mindess & Young, 1981)

GAPS AND CHALLENGES

To reduce energy needs in cement, aggregate, and concrete; to conserve mineral resources; to find durable concrete for use in marine environment and extreme cold weather regions of Canada, long-term research should be undertaken in the following areas:

1. Perform research to optimise grading of cement.
2. Develop processing methodology to produce cement at about 1000°C instead of 1500°C.
3. Develop technical data on supplementary cementing materials such as fly ashes, silica fume, and blast furnace slags to reduce consumption of energy in cement manufacture.
4. Develop processes to obtain byproducts closely resembling cement from fluidized bed processes for the burning of coal.
5. Develop technical databank on non-ferrous slags for use in mine backfill and for low strength concretes in the mining areas.
6. Perform research to utilise aggregates from slags obtained from coal gasification plants.
7. Develop durable concretes for offshore structures.
8. To develop NDT techniques for in situ testing for both above and underwater concretes.
9. Develop technology to manufacture lightweight aggregate with reduced energy requirements.

CERAMICS

Ceramics, by definition, comprise a diverse family of artifacts manufactured from inorganic, non-metallic materials that, at some stage in their fabrication (or end use) are processed at temperatures above red-heat. For the most part, they consist of complex mineral mixtures that undergo a series of physicochemical changes during processing to produce the desired physical, chemical and mechanical properties of the product. Because of variation in the naturally occurring raw materials and the complexity of the compositions, it is rarely possible to precisely predict, on the basis of chemistry and mineralogy, the suitability or processing behaviour of any given constituent. Hence, evaluation of Canadian mineral materials for use in ceramic formulations necessarily involves not only their characterization individually, but also evaluation of their effect on product properties and processing parameters.

For purposes of this report, the ceramic spectrum is divided into five classes: clay products, pottery and whitewares, refractories, glass and glassceramics, and ceramic materials for advanced technology. Each class plays or can be expected to play a significant role in the Canadian context. Less significant classes, e.g., abrasives and porcelain enamels, are included only inferentially.

CLAY PRODUCTS

GENERAL DESCRIPTION OF INDUSTRY

The structural clay products industry in Canada produced goods valued at $127.8 million in 1980, employing a total of 2993 persons, 2350 of whom were directly assigned to production (35). Products include clay building bricks and tiles, sewer pipes, flue-liners and farm drain tiles. Production has remained steady for the past several years, with increasing value partly reflecting inflation. The industry comprises four major multiplant corporations and a continually decreasing number of (usually very small) independent plants.

CURRENT TECHNOLOGY

Raw materials are common clays and shales, usually obtained at or near the plantsite, in turn located near major markets. Processing involves comminution, forming, drying and firing stages: the larger plants use best North American state of the art technology, including stiff-mud extrusion and tunnel-kiln firing.

PROBLEMS AND CHALLENGES

The process is energy intensive. In recent years the industry has significantly reduced energy consumption (reportedly by 15-20%) by closer control of the firing process and better utilization of waste heat; more efficient European tunnel kiln designs have not yet been adopted owing to capital costs. Environmental considerations can be expected to have major effects in the next two decades. Restrictions on quarrying (particularly in central Canada) and emissions (noteably fluorides) will prompt exploration for new deposits and possible relocation of plants. R & D will be required to provide new materials and develop new products, particularly of lighter weight, larger size and increased durability under Canadian climatic conditions.

CURRENT RESEARCH AND DEVELOPMENT

Current Research and Development is performed largely on an individual company basis, usually in concert with quality control; no statistics are available, but expenditures are estimated at about $400-500K annually. In addition, a research program at NRC into durability of bricks has been supported for the past three years on an industry-wide basis ($225K). Some federal government funding has been provided through the Industrial Research Assistance Program, administered by NRC. EMR is proposing to fund R & D into fluoride emissions control and resultant energy inefficiencies, at $825K under the National Energy Program's minerals program.

WHITEWARES AND POTTERY

GENERAL DESCRIPTION OF INDUSTRY

The whitewares and pottery industry in Canada produced goods valued at $83.4 million in 1980 from imported clays: the total workforce is reportedly 1882 persons, with 1523 assigned directly to production (36). Including goods made from indigenous clays, for which no statistics are available owing to the multiplicity of small operators, the total value undoubtedly exceeds $100K. Imports (mostly tableware) in recent years have been steady at about $60 million. The industry presents a great diversity in products and plant size. Products include fine chinaware, tableware, sanitaryware, electrical and technical porcelain, and ceramic wall tiles (which are traditionally manufactured from white-burning raw materials) and artware and floor tiles (for which the underlying body -- exclusive of the glaze -- may be coloured). Plants range from very small to medium in the tableware/artware sectors to medium to large in the electrical/sanitaryware sectors; taken together the sanitaryware producers dominate the industry followed by the tile manufacturers.

CURRENT TECHNOLOGY

Indigenous clays are commonly used in the artware sector, where clay preparation usually involves only washing and screening to remove gross impurities: mechanization and the use of modern, fuel-efficient kilns are usually precluded by the small-scale operations. In the whitewares group, raw materials include clays, feldspar, nepheline syenite, silica, alumina, talc, and minor minerals such as whiting and wollastonite. All materials are purchased to ceramic grade specifications from industrial minerals suppliers. Their preparation includes the usual processes of grinding, sizing, washing/leaching, flotation, magnetic separation, with attendant problems related to energy consumption and environmental concerns: these will be intensified and will require more R & D as existing high-grade ores continue to be depleted. In general, the ceramic manufacturing processes are normal state of the art for North America. Body preparation in the tile industry is partly automated, and employs spray drying; the nature of the product lends itself to mechanical handling and glazing, though mechanical loading and unloading is rare. Because of the diversity and large sizes of various types of sanitaryware, the casting process is employed. Automatic casting is not practised in Canada -- difficulties of control and heavy losses during upsets have discouraged its adoption in North America, although it appears to be the norm in the USSR. Similarly, loading of kiln cars is manual.

PROBLEMS

Production of all ceramics is energy intensive; the whitewares industry has made significant progress in reducing energy consumption by improved kiln controls, changing from muffled to direct-fired kilns, single firing of glazed ware and better utilization of waste heat. Recently, the first roller-hearth kiln firing sanitaryware in North America was commissioned at Eastern Potteries Ltd.; this highly efficient installation was partly funded by EMR's Office of Energy Conservation.

CURRENT RESEARCH AND DEVELOPMENT

The extent of R & D in the Canadian pottery and whitewares industry is not recorded, but is unquestionably small, given the small size of many plants and the fact that the major producers are subsidiaries of foreign companies with research facilities largely centered in the USA. Development of new glazes and improved body formulations is carried out on a routine basis at a number of plants. Some federal funding has been provided through the Industrial Research Assistance Program, contracts usually being of the mini-IRAP type.

GAPS AND CHALLENGES

Increased efficiency is largely possible through increased automation and fuel efficiency to be gained by introducing further kiln technology capable of firing cycles registered in minutes rather than hours. Much R & D will be required to develop bodies and glazes capable of maturation in such fast-firing regimes.

REFRACTORIES

GENERAL DESCRIPTION OF INDUSTRY

A variety of acid and basic refractories, both fired and unfired types, are produced in Canada for use as high-temperature resistant linings for furnaces for heat generation (boilers, heat treating equipment) and processing (pyrometallurgical smelters and refiners, ceramic and cement kilns). Shipments of manufactured products in 1980 were valued at $135.7 million, and until the current recession had been growing at 6-8%/a (37). The establishment of a major new plant (Didier Refractories) in 1979 had a major impact on the value of goods produced. The industry is dominated by 4 or 5 large plants (subsidiaries of foreign parents) and several smaller (some Canadian owned) firms. The former generally provides a more comprehensive range of products, whereas many of the latter make only specialty products, such as castables and mortars, and have no firing facilities. The comparatively small Canadian market for many special types of refractories limits the variety actually produced to those in major demand by Canadian industry or, rarely, for export, e.g., Canadian Refractories Ltd.

CURRENT TECHNOLOGY

Medium- and high-heat duty fireclay brick, primarily used in boiler settings, are produced from both indigenous and imported fireclays, in both dense and insulating varieties: more-refractory types are produced by additions of imported bauxite. Insulating bricks of diatomite are manufactured by one Canadian company. Basic brick, for use in ferrous and non-ferrous metals refining are manufactured from Canadian and imported magnesite and dolomite, along with imported chrome ore. High alumina and mullite refractories are now being manufactured in Canada from Canadian and imported raw materials.

Process technology is essentially state of the art. Comminution and sizing of materials are automated, as is mixing and pressing of shapes but loading and unloading of kilncars are usually manual. Preparation of grog or deadburned magnesia is usually by rotary kiln with heat recovery systems, and kiln controls and waste heat recovery facilities have been upgraded for increased energy efficiency. Reject brick can usually be recycled to grog.

PROBLEMS

The very high temperatures involved in firing refractories make the industry energy intensive. No facilities exist in Canada for manufacture of large, dense, isostatically pressed or fusion-cast blocks (as used for example in glass tanks).

Recent developments in pyrometallurgy and in thermal power engineering have led to higher temperatures and more aggressive ambient conditions requiring new or improved refractories. Similarly, with the end of the current recession and oil glut it is likely that interest will revive in areas such as gasification and liquefaction of coal, where existing refractories have been shown to be little more than adequate. In fact, no new refractory materials available at a commercially acceptable price are in sight, and improvements are likely only through use of less-impure raw materials and progressive development such as has occurred in chrome-magnesia bricks, which have improved from silicate bonded to direct bonded to re-bonded fused grain to fusion-cast products. New combinations of existing raw materials are also possible, as in current development of carbon-impregnated periclase bricks, currently being investigated by Canadian Refractories Ltd. at Laval University with funding assistance from NRC (38). New silicon carbide compositions are expected to find much wider use although existing manufacturing processes present serious emission problems.

CURRENT RESEARCH AND DEVELOPMENT

R & D in refractories in Canada is going forward on a continuous basis at Ecole Polytechnique, Canadian Refractories Ltd., and presently in CANMET. Intermittent research is underway at Laval University, University of Toronto, McMaster University, University of British Columbia and several other manufacturers and users. Total resources committed at any time are estimated to be in the area of $500-600K/a. The work by CRL is partly funded by NRC.

GAPS AND CHALLENGES

Of equal or more importance, reliable test methodologies are lacking for predicting behaviours of refractories in service in most applications, and need to be developed, e.g., non-destructive methods appear promising.

GLASS AND GLASSCERAMICS

GENERAL DESCRIPTION OF INDUSTRY

In 1980, the value of glass manufacture (melted) in Canada was $484.3 million, with 8604 employees, supplemented by a further $260.8 million in shipments of glass products manufacture such as mirrors and tempered glass, employing a further 3439 persons (39). Container production has remained relatively stable, but flat glass production has been adversely affected by the slump in auto and housing production. Except for a few small operators producing hand-blown artware, all the plants are of substantial size commensurate with economies of scale. The container segment comprises two multi-plant corporations with 12 plants located from coast to coast; two major corporations form the flat glass segment, with plants located in central Canada near the major markets. The third segment, glass fibre, is dominated by Fibreglass Canada Ltd. with four plants, although others produce glass and mineral wools. There is no production of glassceramics or fusion cast materials in Canada.

CURRENT TECHNOLOGY

The raw materials for glass melting comprise silica, nepheline syenite, feldspar, soda ash, salt cake, limestone, dolomite, borax, barytes, and other minor ingredients such as decolourizers. Raw materials are purchased at glass-grade specifications from industrial minerals processors, who beneficiate them by methods standard to the industry. The nepheline syenite and much of the silica, soda and lime are derived from indigenous Canadian sources, the remainder is imported from the USA.

Technologies in glassmaking are North American state of the art. Batching is automatic, frequently computer-controlled; considerable R & D has been done to develop batching techniques to speed melting. Except for fibreglass, which is all-electric, the energy-intensive melting process uses natural gas in regenerative furnaces and increased insulation is being used in furnace construction. Closer combustion control has been instituted to save fuel. Containers are formed by modern, high-speed, multi-station machines. One flat glass producer employs the Pilkington float process, the other a fully modernised Pittsburgh process.

PROBLEMS AND CHALLENGES

Problems in the glass industry relate to needs to improve energy efficiency, increase durability of tank refractories, reduce emissions, develop improved mould materials, recycle used glass and dispose of waste products. Particulate emissions are controlled by dust collectors but emissions of No_x and sulphurous gases are a problem. A current issue in the USA is disposal of used chrome-bearing refractories from the regenerators, considered a source of toxic wastes: research is required into means, preferably, of recycling these materials. Used glass, a significant source of litter, is a beneficial raw material, each percentage of cullet reducing fuel requirements by 0.25 to 0.5%. A recycling program by the Canadian container industry has demonstrated successful reclamation of waste glass, as well as potential use of cullet up to 80% of the batch. Collection now presents the prime problem.

Because glass is an inherently brittle material, its full potential cannot be exploited as a structural material. Research is required into glass failure mechanisms and methods of strengthening it, possibly reinforcing with refractory fibres or plates or development of intrinsic two-phase systems by exploiting immiscible phases.

CURRENT RESEARCH AND DEVELOPMENT

R & D in Canada in glass is limited and strongly oriented to "development" of in-plant processes. Some industry sponsored research on strength of sheet is underway at ORF and some in-house work on glass structure is being conducted at U.B.C. Total effort can probably be valued at several hundred thousand dollars.

CANMET has sponsored contracts ($75K, 2.5 years) to develop glass ceramics for abrasion-resistant linings, currently utilizing waste steel plant slags: this technology is widespread in the USSR and Warsaw Pact countries to produce structural components and abrasion/corrosion resistant materials from waste glass and slags (sitals), as well as fusion-cast basaltic compositions.

CERAMIC MATERIALS FOR ADVANCED TECHNOLOGY

GAPS AND CHALLENGES

A great variety of ceramic materials are under development or envisioned to be required as critical components in devices and systems for energy conservation, conversion and storage, and in the areas of transportation and communications, as well as industrial applications. Most of this product development is at the leading edge of technology expected to reach fruition only in the next 5-20 years, following which much research can be expected to improve performance in the usual process of evaluation of prototypes.

The raw materials required are most often chemically prepared and are of high purity. Processing often involves preparation of precursor powders by chemical precipitation, sol-gel, spray-drying, freeze-drying, etc., product fabrication by conventional forming and sintering or new technologies such as hot isostatic pressing or completely novel process modifications developed to suit the occasion. Problems relate chiefly to optimizing compositions and processing methods to suit the requirements.

Internationally, the effort expended in R & D in these areas is extremely large: in Canada development of these systems undoubtedly is counted in the hundreds of millions of dollars, with the ceramic component probably in the 5-20% range. Much of the R & D is industrial-confidential or has military connotations and thus cannot be determined.

The CANMET program to develop fast ion-conducting ceramic electrolytes for storage batteries and fuel cells (40) is currently provided with about 5 PY and $200K, with possible increase during the 1982-83 fiscal year by 2 PY and up to $500K. Complementary support from NRC and DREE has been in the order of $200K/a. Opportunities are seen to expand this program to develop materials for MHD systems, electrolyzers, therm electric generators (for conversion of waste heat) and a variety of sensors and potential materials (metals) refining systems.

Recently the Japanese announced the first prototype ceramic engine in the diesel configuration; it reportedly employs low-expansion glass-ceramics in the high-temperature components. Over the past several years a great effort has been put forward (largely coordinated by IEA) in developing a high-temperature turbine engine that will employ ceramic SiAlON materials in the rotors and stators. Although some research on processing and engineering properties of both types of materials has been done at Canadian universities, Canada's contribution in this challenging area has been insignificant, but future opportunities will exist for a contribution to improve materials in the normal process of development.

There is a clear need for refractory materials with high thermal-shock resistance for use in high-temperature heat exchangers. There is present and future opportunity to exploit Canadian materials for thermal storage media, particularly in the residential area in conjunction with active solar systems or utilization of off-peak electricity.

There is continuing need for new and improved ceramic electronic components: transducers, semi-conductors, capacitors, substrates suited to higher temperatures and greater heat-sink capacities for use in ultra-high frequency devices, e.g. for communication satellites. Semi-conductors for photogalvanic applications are being researched in Canada, but a vastly increased effort will be required if Canada is ever to become a prime mover in the area of photovoltaics on the international scene.

A number of ceramic materials have potential biological uses, e.g., for implanted prostheses but development is slow and an increased effort could be advantageous.

There has been considerable interest in prognosticating opportunities for ceramic research now and in the next decade. Regardless of origin there is a significant concurrence on major themes, most of which have been touched upon above. A useful overview may be gained from references 41,42,43.

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APPENDIX - DIRECTORY OF FEDERALLY SUPPORTED RESEARCH IN UNIVERSITIES

1973-1974

1. Crystal chemistry of phosphate minerals. 2. Crystal chemistry of zirconium minerals; Chao, G.Y.; Geology; Carleton University, Ottawa, Ont.; National Research Council of Canada; $11 900

1972-1973

1. Crystal chemistry of phosphate minerals. 2. Crystal chemistry of zirconium minerals; Chao, G.Y.; Geology; Carleton University, Ottawa, Ont.; National Research Council of Canada; $8500

1971-1972

Synthesis and stability of MG-FE-CA-Pyroxenes and Tritium-Labelled asbestos minerals; Turnock, A.C.; Earth Sciences; the University of Manitoba, Winnipeg, Man.; National Research Council of Canada; $7200

LIGHT AND REACTIVE METALS

This commodity grouping includes the "light metals", aluminum and magnesium, and the "reactive metals", titanium and zirconium. The term reactive metal is applied to titanium and zirconium because they have a strong affinity for gases in the air at elevated temperatures and, when combined, form very stable compounds. Although not related chemically or in terms of common extraction processes, they are included with the light metals for convenience.

ALUMINUM

GENERAL DESCRIPTION OF INDUSTRY

The world aluminum industry is essentially controlled by six multinational companies: Alcan, Alcoa, Kaiser Aluminum, Reynolds Metals, Pechiney Ugine Kuhlmann (PUK), and Alusuisse. These companies are fully integrated and own sometimes on a shared basis, the mines, alumina refineries, smelters and fabricating plants of the world. They control nearly 50% of the world capacity in this metal and are associated with practically all new projects of international significance (1). In Canada, metal production of approximately 1 Mt/a is accounted for by Alcan (85%) and Reynolds (15%) with smelters in Quebec and British Columbia. With the present recession all the world's smelters are operating at reduced capacity (Canada 88%, USA 65%). In 1980, Canada exported 784 720 t valued at $1500 million.

Canada's low electrical energy costs, principally hydroelectric, relative to the rest of the world give it a strong position as a major aluminum producer. Japan, which a few years ago was the third largest producer of aluminum (a position now held by Canada), has been forced to cut production drastically because of its high energy costs (2). For this same reason it is unlikely that many new smelters will be built in the USA. Future smelting will go where there is cheap power and in fact Canada's smelter capacity has been recently increasing. The production of alumina from bauxite will tend to become more concentrated in the countries which have the ore. One reason being that it is cheaper to transport alumina than bauxite.

Aluminum is the most abundant metallic element in the earth's crust, 8.8%. Its ore, bauxite, is spread throughout the world principally in countries of temperate climate. In 1980, Canada imported 3.5 Mt bauxite and 984 000 t alumina, principally from Australia, Africa, South America and the Caribbean; these operations generally are owned or partially owned by Alcan.

CURRENT TECHNOLOGY

In the Bayer process, which transforms bauxite to alumina, bauxite is digested with sodium hydroxide to dissolve alumina, which is purified, precipitated as a trihydrate and calcinated to Al_2O_3. This alumina is the feedstock to the Hall-Hérault process wherein it is electrolyzed in a fused salt bath of sodium-aluminum fluoride to metallic aluminum. Although both processes are approaching 100 years, they are used to produce almost all the world's aluminum. They are being continuously improved and automated principally to increase production capacity and to reduce the high energy consumption. Examples are flash fluid calciners replacing rotary kilns, improved cell and anode design, addition of LiF to the cell and computer controls. Cell current efficiency is now approaching 95% and power requirements to reduce the oxide to the metal have been lowered by about 30% in the past 30 years to 13 kwh/kg (3). These improvements, as well as the use of alumina scrubbers to capture fluoride emissions from the cells, have reduced atmospheric pollution and improved the working environment.

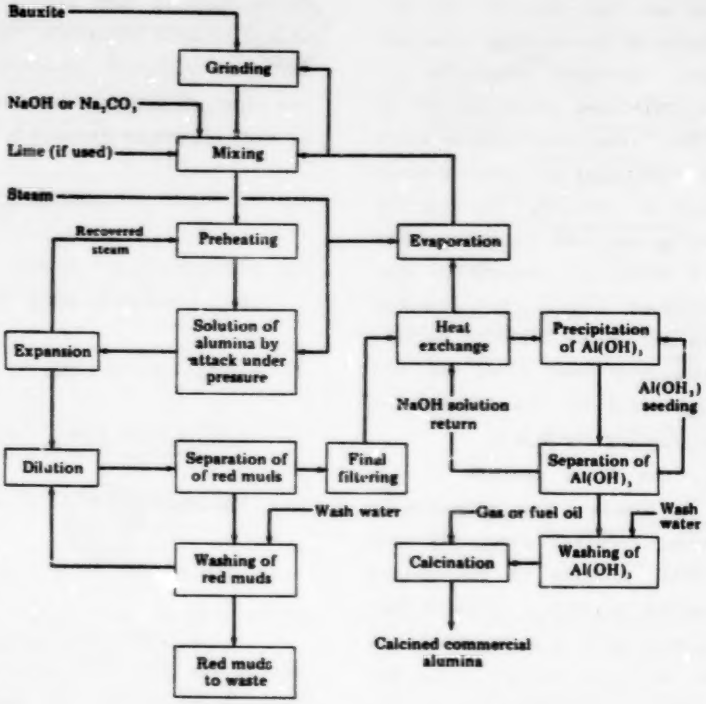
Bauxite
Grinding
NaOH or Na_2CO_3
Lime (if used)
Mixing
Steam
Recovered steam
Preheating
Evaporation
Expansion
Solution of alumina by attack under pressure
Heat exchange
Precipitation of $Al(OH)_3$
$Al(OH)_3$ seeding
NaOH solution return
Dilution
Separation of of red muds
Final filtering
Separation of $Al(OH)_3$
Wash water
Gas or fuel oil
Wash water
Washing of red muds
Calcination
Washing of $Al(OH)_3$
Calcined commercial alumina
Red muds to waste

Fig. 8 - Flowsheet of the Bayer process

PROBLEMS

The aluminum industry is a mature industry, but it is continuously ameliorating its processes to improve the environment and reduce costs. The technology for reducing fluoride emissions with alumina scrubbers is in place in many smelters. Red mud tailings are still impounded although numerous R & D attempts have been made to recover the residual alumina and soda content. The energy required is a very large factor in aluminum smelting and is the object of continuous R & D and process development. Up to 18 000 kwh is required per tonne of aluminum compared with 150 kwh for steel. As mentioned, Canada must import all its raw material, bauxite and alumina, thus raising the question of security of supply. However, bauxite deposits are so widely dispersed throughout the world that lack of supply is not a major threat.

CURRENT RESEARCH AND DEVELOPMENT

In 1974, the formation of the International Bauxite Association (IBA) as a cartel spurred research by many countries into non-bauxite alumina sources and processes. CANMET had its own project investigating the resource base of alumina-bearing clays, coal wastes, and anorthosite. Various processes were investigated to extract alumina from these materials and cell grade alumina was produced. However, the processes were more complex and were evaluated as being more expensive than the Bayer process using bauxite. As a result of CANMET's work and discovery of an acid soluble anorthosite deposit, Alcan then worked on and refined the acid-anorthosite process. They are still evaluating their results. The USBM, with direction and partial funding by a number of aluminum companies (including Alcan and Reynolds; CANMET conducted some of the research required and had observer status), built several miniplants to investigate non-bauxite alumina processes. The most attractive was an HCl-clay process. This work continues but at a lower level of effort. Alcan and Pechiney built and operated a 20 t/d H_2SO_4-clay or shale process (H^+ process) in France in 1975-76. The results are still being evaluated and some laboratory work continues. The R & D effort on non-bauxite alumina has been decreased lately for various reasons: the IBA did not become a strong cartel, large bauxite deposits have been found and are being developed in Australia and Brazil, and the recession has reduced funding for R & D.

In smelting, the Alcoa process is the most recent attempt to displace the Hall-Hérault process. It reportedly requires about 30% less power. In the process, alumina is impregnated with oil and chlorinated to $AlCl_3$. This is then electrolyzed in a fused salt bath to metal. A 15 000 t/a plant was built in the mid 1970s and plans to have it expanded to 50 000 t/a in the late 1970s, did not materialize. Technical problems still exist, probably associated with the chlorination step.

Alcan has two R & D laboratories in Canada at Arvida, Quebec and Kingston, Ontario, one each in the UK and Spain. Also, each smelter is involved in process R & D. Including headquarters staff associated with R & D, technology and engineering Alcan had 1200 persons and a budget of $47 million in 1980 (2). Reynolds R & D labs are in the USA. Some small scale research is occasionally carried on at various universities. As mentioned, CANMET was involved in an alumina project from 1975-80. All the aluminum companies in North America have been historically very secretive about their R & D efforts. Such secrecy makes any R & D CANMET might contemplate less effective. Alcan has been and would be the principal beneficiary of CANMET's work.

GAPS AND CHALLENGES

Aluminum is a $1000 million Canadian industry thus CANMET should maintain a knowledge base in alumina-aluminum. CANMET could contribute by maintaining some effort on the resource base of non-bauxite materials for security of supply. Our wider viewpoint of extractive metallurgy than the aluminum industry has, may in future indicate an area of useful process R & D.

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MAGNESIUM (CALCIUM, STRONTIUM)

GENERAL DESCRIPTION OF INDUSTRY (1,2)

In 1980, world production of primary magnesium was 320 000 t with the USA accounting for about half of this and the USSR and Norway 24% and 15% respectively.

In 1980, the production of magnesium in Canada was 9200 t with a value of $27 million of which 6000 t was exported. Canadian consumption was 5000 t which included 3500 t of imports.

Commercial-grade magnesium, which represents the major proportion of Canadian production, is used for general fabrication and alloying, mostly with aluminum to impart hardness and strength. Magnesium is increasingly used in the ferrous industries as a deoxidizing and desulphurizing agent. It is also used as a reducing agent in the production of titanium and other reactive metals as well as or cathodic protection.

CURRENT TECHNOLOGY

Primary magnesium is produced by two basic processes. The first is by electrolysis of magnesium chloride derived from seawater and brines. The second, the Pidgeon process reduces magnesium from ores such as dolomite or magnesite, with ferrosilicon at high temperatures. Flowsheets for the two methods are shown in Fig. 9 and 10 (3). Canadian production, as well as plants

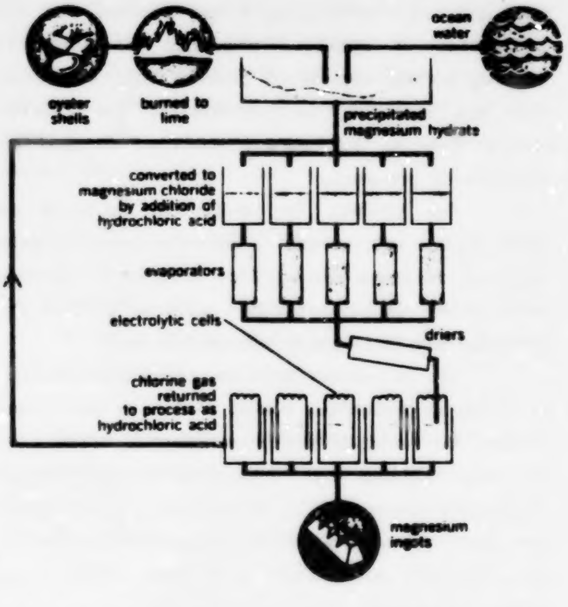
oyster shells
burned to lime
ocean water
precipitated magnesium hydrate
converted to magnesium chloride by addition of hydrochloric acid
evaporators
electrolytic cells
driers
chlorine gas returned to process as hydrochloric acid
magnesium ingots

Fig. 9 - Diagram showing the electrolytic extraction of magnesium from sea water

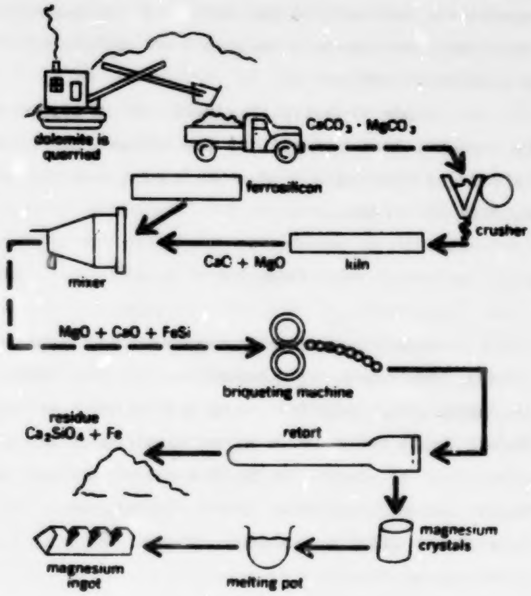
dolomite is quarried
$CaCO_3 \cdot MgCO_3$
ferrosilicon
crusher
mixer
CaO + MgO
kiln
MgO + CaO + FeSi
briqueting machine
residue
Ca_2SiO_4 + Fe
retort
magnesium crystals
magnesium ingot
melting pot

Fig. 10 - Diagram showing extraction of magnesium metal from dolomite ore by ferro-silicon process

in France, Italy, Japan and one in the USA, use the latter. The other producers use the electrolytic method. The electrolytic process consumes considerably more energy than the ferrosilicon reduction process.

The Chromasco Ltd. plant at Haley, Ontario which is the only Canadian producer of primary magnesium, is rated at 11 000 t/a (1). Calcined dolomitic limestone ($MgCO_3.CaCO_3$) is mixed with ferrosilicon and compacted into briquettes which are charged into horizontal retorts eleven 11 in. diam and 10 ft long within the hot zone (1150-1200°C) of a gas or electrically heated furnace (4). The retorts are evacuated and as the reaction proceeds, magnesium vapour condenses in the water-cooled end of the retort which is outside the furnace.

Magnesium is also commercially electrowon from a $NaCl-KCl-MgCl_2-CaCl_2$ fused salt electrolyte, maintained at 700-790°C, which is more dense than the magnesium metal itself (5). For this reason, in commercial magnesium chloride electrolysis cells (the Dow and I.G. Farben (IGF) cells), the cell cathodes are equipped with collection systems, which trap the rising metal and channel it outside the cells.

The Dow cell is designed to operate with a magnesium chloride feed that contains 25 wt % water, whereas the IGF cell operates with an anhydrous liquid $MgCl_2$ feed. In the Dow cell, water vapour evolved at the introduction of the hydrous $MgCl_2$ feed reacts with some of the chlorine evolved at the anode to form gaseous HCl. Since no attempt is made to recover concentrated chlorine gas from the anodes, the anode-cathode spacing is made as small as possible consistent with preventing the back-reaction of magnesium and chlorine. In practice, this anode-cathode separation is 4 cm, which allows high current densities to be achieved.

The gaseous mixture of HCl and Cl_2 is converted in an acid furnace essentially to hydrochloric acid which is then used to dissolve $Mg(OH)_2$ to produce $MgCl_2$ solutions for crystallization.

Since feed to the IGF cell is anhydrous $MgCl_2$, concentrated chlorine is evolved at the anodes and recovered for chlorination of magnesium oxychloride-coal-peat briquettes. An anode-cathode spacing of 13 cm is necessary for chlorine recovery, however, and this increases cell resistance and thus decreases current density. The production rate of the Dow cell is 20 kg magnesium per hour, that is, a similar output to that of the Hall-Héroult cell.

A new prototype Dow cell, which apparently uses 50% less energy than the present Dow cell, is being tested with a view to employment for expansion of capacity.

PROBLEMS

Among the recent pressures and problems facing the magnesium production industry today, it appears that the following are of primary concern. Until recently, a major requirement for the production of magnesium by the retort method was heavy manual exertion under significant heat stress and dust conditions during charging and discharging (4).

Another concern within the industry is the energy consumption for the production of magnesium (4).

The lack of a Canadian source of strontium minerals (celestite $SrSo_4$, and strontianite $SrCO_3$) appears to be of concern (4), since Chromasco produces 100 tonnes of strontium per year by aluminothermic reduction of SrO, mostly for alloying with aluminum, and must import its raw feed materials. Concern was also expressed over the availability of relatively high grades of limestone, since these are required for the 500 t/a of aluminothermic calcium production.

CURRENT RESEARCH AND DEVELOPMENT

The problem of heavy manual labour in an unpleasant working environment is being overcome by the implementation of hydraulically-operated pneumatic charge-discharge systems which provide a very significant reduction in the work load and greatly reduce the risk of accident and health problems to workers (4).

The Société Nationale de l'Amiante (SNA) which is the Quebec crown corporation empowered to enter fields related to asbestos, has recently developed and operated a pilot plant to produce pure $MgSO_4$ from $MgSiO_3$-bearing asbestos tailings, of which 20-30 Mt/a are produced in Quebec (6,7). They are also developing a new process for the production of metallic magnesium (8) which is related to the magnetherm process but is simplified and intended, in contrast to the Pidgeon retort process, to operate continuously and at atmospheric pressure.

The process for magnesium sulphate is based on the calcination of asbestos tailings with ammonium sulphate and will be adapted to produce $MgCO_3$ which in turn may be calcined to form MgO feed for magnesium metal production (8).

It is evident that the above innovations, if taken to production scale, will have beneficial results on reducing the environmental effects of asbestos tailings and recovering valuable metals from waste products. It is too early to determine whether the new process for magnesium production will be more energy efficient than the conventional processes.

Research on the thermodynamics and kinetics of the direct reduction of MgO is presently being conducted at the University of Waterloo (2) with one of the objectives being to reduce the energy requirements by recovering some of the enthalpy of condensation of the magnesium vapour.

Research is also being conducted at Chromasco (4) into the development of a new, more thermally efficient electric furnace than those currently being employed. Additionally, modifications are being made to their present furnaces.

High purity dolomite availability is not expected to be a problem in the future.

Expansion of production by overseas suppliers appears not to be of concern since the plant is operating at about 75% of capacity.

Environmental issues are related to the non-toxic calcium silicate dust emissions from the retort residues. Through the use of bag-houses and scrubbers, these emissions are minimized.

GAPS AND CHALLENGES

Major improvements in the work load associated with the retort production of magnesium have been effected in the last decade. Reduction of energy consumption by the retort method should continue as a primary consideration in research on magnesium production. Little other significant change to the basic retort process is anticipated.

The development of quality sources of limestone, strontianite and celestite are of concern to the Canadian producer of calcium and strontium.

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TITANIUM

GENERAL DESCRIPTION OF INDUSTRY

Although titanium dioxide has been used for pigments and fillers for over 60 years, the metal has been produced commercially for only half that time. Even now only 10% of titanium ores are used for metal production of which about 60% goes to the aircraft and aerospace industry (1). In 1980, approximatley 100 000 t of titanium metal sponge was produced by the USSR, USA, Japan, UK and China (2). There is no metal production in Canada.

The free world sponge capacity should exceed 56.7 Mkg in 1982, an increase of 4.3% over 1980. The production of the metal was initially sponsored by the US military for aerospace and has fluctuated according to their needs, e.g., one B-61 bomber would use 90 720 kg of metal (3). However, because of the metal's light weight, high strength and excellent resistance to corrosion and chemicals, non-aircraft uses are becoming more common. In Russia, titanium is often used where we would use stainless steel. They recently built a submarine using titanium which resulted in their reducing exports. This, combined with high demand, forced the price from $1.47 to about $11.34 kg. It is now about $2.27/kg (4).

Canada does have very large reserves of ilmenite ore and one company, QIT-Fer et Titane, Quebec, produces an ilmenite concentrate (2.5 Mt/a capacity) and a titanium slag (860 Kt/a capacity) (5). The ilmenite ore is upgraded from about 86 to 93% combined titanium and iron oxides by heavy media, spirals and cyclone separators. In 1979, Canada was the world's third largest exporter of ilmenite concentrate. QIT also produces Sorelslag (70-72% TiO_2) and Sorelmetal (pig iron) by first calcining the ilmenite concentrate to remove sulphur and then smelting it in electric furnaces with coke. QIT is planning improvements to its electric furnaces and auxiliary equipment. About 90% of the slag is exported. The remaining 10% is sold to two Quebec producers of TiO_2 pigment. This is made by sulphuric acid leaching of the slag, filtration to remove iron and silica, seeding, precipitation and calcining of TiO_2. Both producers have the same 36 Kt/a capacity. All three of the above companies are foreign owned (5).

There are two small titanium metal fabricators, the larger one being Titanium Ltd/Ltée with annual sales estimated at $10 million (1982). They import sponge (which is melted by Eldorado Nuclear), ingots and plates which are rolled by Atlas Steels Ltd. to various shapes used by Titanium Ltd/Ltée to fabricate heat exchangers, tubes, etc. This company had investigated building a sponge plant in 1979-80. Its cost at $50-60 million and lack of government support (in fact difficulties with FIRA) cancelled the project. It would have operated on imported rutile (TiO_2) (4).

CURRENT TECHNOLOGY

Titanium sponge is produced by the Kroll process. A TiO_2 concentrate of rutile, ilmenite or slag is chlorinated with coke to $TiCl_4$. The tetrachloride is then reduced to sponge metal with magnesium or sodium metal in a closed inert system. It is a batch operation and although some reactors have been increasing in size such that 3-7 t of sponge can be produced per batch, most are kept small in order to keep good quality control on metal. Residual magnesium is removed by vacuum distillation or by leaching; residual sodium is leached. The sponge is formed into consumable electrodes with scrap or alloy and melted in a vacuum arc furnace and cast as large ingots.

Another process, developed in the USA and Japan, but not yet commercialized on a large scale, is the electrolysis of $TiCl_4$ in a molten salt bath (2,6). Shawinigan Chemicals operated a pilot plant from 1948 to 1951 using molten salt bath electrolysis. The D-H Company announced the operation of its 1 M lb electrowinning plant in 1981. It shut down in early 1982 due to technical problems (7).

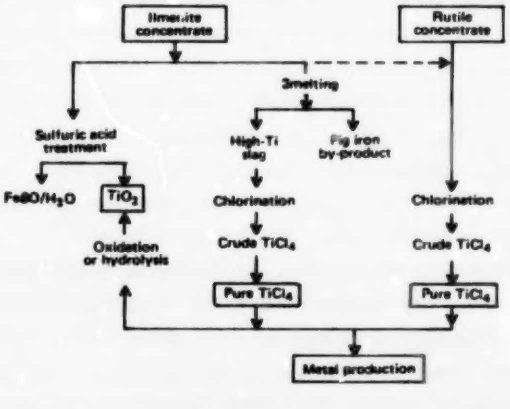
Ilmenite concentrate
Rutile concentrate
Smelting
Sulfuric acid treatment
High-Ti slag
Pig iron by-product
FeSO/H₂O
TiO₂
Chlorination
Chlorination
Oxidation or hydrolysis
Crude TiCl₄
Crude TiCl₄
Pure TiCl₄
Pure TiCl₄
Metal production

Fig. 11 - Conversion of ilmenite and rutile to refined TiO_2 and $TiCl_4$. Courtesy American Society for Metals

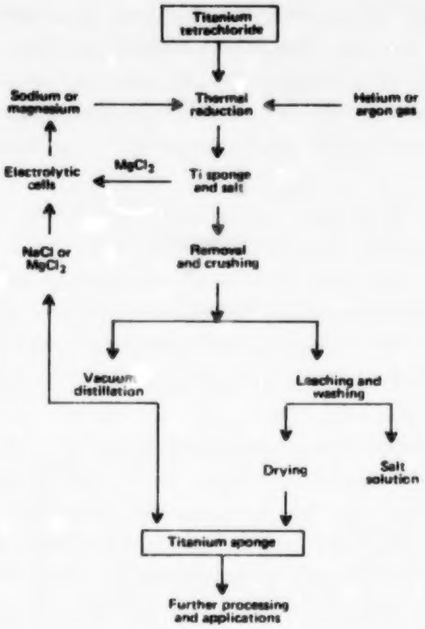
Titanium tetrachloride
Sodium or magnesium
Thermal reduction
Helium or argon gas
Electrolytic cells
MgCl₂
Ti sponge and salt
NaCl or MgCl₂
Removal and crushing
Vacuum distillation
Leaching and washing
Drying
Salt solution
Titanium sponge
Further processing and applications

Fig. 12 - Thermal reduction of titanium tetrachloride to sponge titanium. Courtesy American Society for Metals

PROBLEMS

A major problem affecting the titanium industry is its lack of stability, that is, the extreme variation of supply and demand as a result of being so closely tied to the aerospace industry (8). Presently there is a large excess capacity. Energy requirements to make the metal are high, about one and a half times that of aluminum. The raw material from which this metal is usually made is rutile which would have to be imported, although it can be made from ilmenite.

CURRENT RESEARCH AND DEVELOPMENT

The only R & D in Canada appears to be by a private organization, Titane de St-Laurent which is doing research with a group of UK scientists (9). It has a pilot plant in England and claims to have produced metals from uncalcinated TiO_2. A pilot plant is to be built in Ottawa and then a plant in Quebec. Sorelslag will be the feed stock.

It is an electrochemical process but apparently not based on chloride metallurgy. It was decided to keep the process secret rather than to patent it. There is a lot of interest in the USA, e.g., Timet, Boeing and also in Canada but people are waiting to see metal produced.

GAPS AND CHALLENGES

An area of research which QIT and at least one pigment plant are working on is HCl leaching of Sorelslag to make TiO_2. Presently this is done with H_2SO_4 but there is an environmental problem with the waste acid which must be neutralized (7). CANMET could possibly assist with this problem.

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ZIRCONIUM

GENERAL DESCRIPTION OF INDUSTRY (1,2)

In 1980, world production capacity of zirconium was 7100 t of which Teledyne Wah Chang Albany (TWCA), Pechiney Ugine Kuhlman (PUK) and Western Zirconium Inc. (WZI) accounted for 92%. Western world consumption of zirconium was estimated at 3300 t for nuclear power stations and 900 t for non-nuclear applications. Although world zirconium production is relatively small, it is nonetheless critical in that zirconium is the only metal suitable for cladding of nuclear reactor fuel bundles. Reactor grade zirconium accounted for a "substantial portion" of Canada's 278 t of 1980 imports of zirconium metals and alloys.

Australia is the major world supplier of zircon, which is imported into Canada (27 500 t in 1980) to make cores and facings on foundry moulds.

Zirconia (ZrO_2) is also imported, usually in the form of baddekyite (64 t in 1980) to make artificial abrasives by fusion with alumina (Al_2O_3).

Ferrozirconium alloys are imported (232 t in 1980) for use in corrosion resistant vessels and piping.

CURRENT TECHNOLOGY (3,4)

The conventional process for the production of reactor grade zirconium is influenced by the necessity of separating hafnium and zirconium which are practically indistinguishable in chemical properties and always occur together; there is about 2% hafnium present in zirconium. The separation is required since the presence of hafnium in the zirconium cladding of uranium fuel bundles would poison the nuclear reaction due to the high cross section of hafnium for thermal neutrons.

As shown in Fig. 13, reactor grade zirconium is conventionally produced as follows:

(1) Zircon sand ($ZrO_2.SiO_2$) is heated with coke in an arc furnace to produce zirconium carbonitride and volatile SiO.

(2) The zirconium carbonitride is treated with chlorine to produce zirconium tetrachloride which is purified of oxides and other chlorides by resublimation in an inert atmosphere.

(3) Hafnium is separated from the zirconium at this point by dissolution of the crude zirconium tetrachloride in ammonium thiocyanate solution and solvent extraction of the hafnium thiocyanate complex by methyl isobutyl ketone (MIBK). The zirconium remaining in solution is precipitated with $NH^{4}OH$ as the hydroxide and kilned to produce ZrO^{2}.

(4) The hafnium-free zirconium oxide is then pelletized with carbon and powered sugar and reacted with chlorine a second time to produce a pure $ZrCl_4$ sublimate.

(5) The $ZrCl_4$ is reacted with liquid magnesium under argon to form sponge zirconium and by-product magnesium chloride which is separated from the sponge by melting and vacuum distillation.

(6) The zirconium sponge is crushed, pressed into bars, arc-melted under vacuum and cast into ingots.

Production of other grades of zirconium is conducted by omitting steps 3 and 4.

PROBLEMS

Zirconium is a strategic requirement for the CANDU reactor system, but is not produced in Canada. In the event that domestic supplies of reactor-grade Zr should be required, it would be necessary to develop Canadian production facilities. Canadian imports in 1980 amounted to about 300 t with a value of about $13 million (1).

CURRENT RESEARCH AND DEVELOPMENT (5,6,7)

In 1977, an ad hoc committee with representation from Ontario Hydro, Noranda Mines, Universities of Toronto and Waterloo, the Province of Alberta, Eldorado and CANMET was formed as a result of concerns expressed by Ontario Hydro over the long-term security of supply of CANDU-grade zirconium (5). The committee was to coordinate the development of new Canadian technology for the production of hafnium-free zirconium and in fact NSERC funding for this purpose was awarded to certain of its members.

A subsequent Ontario Hydro study showed that a Canadian plant producing zirconium by conventional methods could be justified only if it were of such a size that half of its production

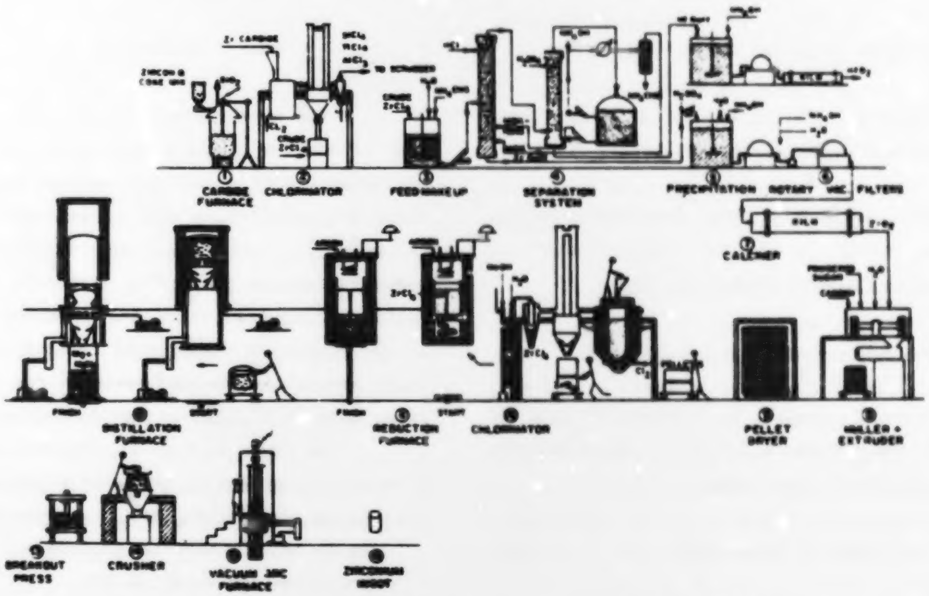
ZIRCON & COKE MIX
Zr CARBIDE
TO SCRUBBER
① CARBIDE FURNACE
② CHLORINATOR
③ FEED MAKEUP
④ SEPARATION SYSTEM
⑤ PRECIPITATION
⑥ ROTARY VAC FILTERS
KILN
⑦ CALCINER
⑧ PELLET DRYER
⑨ MULLER + EXTRUDER
CHLORINATOR
REDUCTION FURNACE
DISTILLATION FURNACE
BREAKOUT PRESS
CRUSHER
VACUUM ARC FURNACE
ZIRCONIUM INGOT

Fig. 13 - Conventional production of reactor-grade zirconium

could be exported (6). However, with a more than adequate supply of zirconium from the present suppliers who, due to the postponement or termination of many nuclear power programs, are operating at about 70% capacity, there appears to be no urgent move towards self-sufficiency in zirconium production. Furthermore, the appearance of the second US supplier (WZI) on the market appears also to have reduced Ontario Hydro's level of interest in developing a Canadian supplier of zirconium (7).

This picture could change in the event that new Canadian technology could produce reactor grade zirconium at costs significantly lower than those of conventional technology.

At present, there appears to be no research being conducted in Canada on zirconium extraction, although zircon occurs to the extent of about 0.03% in Alberta tar sand (1) and may be extracted in the form of concentrates by physical methods (8). However, research has recently been conducted at the University of Waterloo and at the University of Toronto.

The Waterloo project was directed towards developing a continuous process for the distillation separation of hafnium from zirconium in chloride systems. Some success was achieved with a reboiler-condenser system and NaCl as the carrier salt (7). However, work was discontinued before conclusive results could be obtained. The effort amounted to $186 000 and 4 PY (9).

The University of Toronto project showed that zirconium and hafnium could be separated by the vapour phase reaction of the impure zirconium tetrachloride with various binary liquid alkali chlorides. Their maximum separation efficiency was about 48% (10).

It is evident from the description of the conventional process for producing nuclear grade zirconium, with duplicate chlorination steps and an expensive solvent extraction state (11), that a challenge exists to develop a Canadian process for reactor grade zirconium production. A new simplified process for the production of nuclear grade zirconium has been proposed by the University of Waterloo (3).

The process involves the chlorination of zircon with calcined petroleum coke in an electrothermal fluid bed chlorinator, separation of silicon tetrachloride from the $HfCl_4$-bearing $ZrCl_4$ vapour by fluid bed condensation and separation of $HfCl_4$ from $ZrCl_4$ by reflux distillation from NaCl-KCl eutectic solution. The $HfCl_4$-$ZrCl_4$ separation is based on the vapour pressure differential between $HfCl_4$ and $ZrCl_4$ in the NaCl-KCl eutectic at about 400C. The remainder of the proposed process is essentially similar to the conventional one, except that magnesium and chlorine are recovered from the magnesium chloride byproduct by fused salt electrolyses.

For the production of zirconium for non-nuclear applications, the process would be further simplified by the omission of the distillation separation of Hf from Zr.

While the process appears to be for the most part feasible, most of the stages require verification on a pilot scale.

GAPS AND CHALLENGES

It would appear that while there is no urgent requirement for the domestic production of nuclear-grade zirconium, it is uncertain how long the present stable supply situation will last. Given the strategic importance of zirconium to the CANDU reactor system, a case could be made for the development of a Canadian process to the pilot stage.

Zirconium-bearing minerals, the most important of which is zircon ($ZrO_2.SiO_2$) are not produced in Canada, so that all domestic requirements as well as metallic zirconium are imported (1).

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APPENDIX - DIRECTORY OF FEDERALLY SUPPORTED RESEARCH IN UNIVERSITIES

1979-1980

Dissolved AL, silicon dioxide and LI in marine sediment pore waters: evidence for clay mineral authigenesis; Moore, R.M.; Oceanography; Dalhousie University, Halifax, N.S.; Natural Sciences and Engineering Research Council; $28,000

1974-1975

1. Crystal chemistry of zirconium minerals 2. Descriptive and comparative studies of rare minerals from nepheline syenite, Mt. St. Hilaire, Quebec; Chao, G.Y.; Geology; Carleton University, Ottawa, Ont.; National Research Council of Canada; $7900

1971-1972

1. Crystal chemistry of phosphate minerals. 2. Crystal chemistry of zirconium minerals; Chao, G.Y.; Geology; Carleton University, Ottawa, Ont.; National Research Council of Canada; $8500

1. Fluid-bed chlorination of zircon. Direct separation of $ZrCl_4$ from $HfCl_4$ 3. Liquid-liquid extraction studies 4. Metal reduction studies 5. Electrowinning of hafnium; Spink, D.R.; Chemical Engineering; University of Waterloo, Ont.; National Research Council of Canada; $7000

URANIUM

GENERAL DESCRIPTION OF INDUSTRY

There are five primary uranium producers in Canada: Denison Mines Limited, Rio Algom Mines Limited, Cluff Mining (Amok Ltd./Saskatchewan Mining Development Corporation-partnership), Gulf Minerals Canada Limited (partnership with Uranerz Canada Limited and Eldorado Nuclear), and Agnew Lake Mines Limited. Rio and Denison operate underground mines, Cluff and Gulf operate open pits and Agnew Lake is a surface heap-leaching operation. During the past year the economic recession and decreased world demand for uranium has resulted in Eldorado Nuclear closing its Beaverlodge operation and Madawaska Mines closing at Bancroft. In addition, ESI Resources in Calgary, after less than two years operation for the recovery of uranium as a byproduct of phosphoric acid, has ceased production.

Production figures for 1980 when all the above plants were in operation, amounted to 7145 t U (1). About 67% of Canada's total uranium shipments in 1980 were from the Ontario mines with the remainder from Saskatchewan. An estimated 6368 t U (90% of Canadian production) valued at $638 million was exported in 1980. In spite of a general slow-down in the industry over the past two years, mill expansion continued at Denison Mines and Rio Algom Mines in Elliot Lake.

A list of operating and planned ore processing plants are given in Table 13. As of December 1980, the total number of persons employed in Canada's uranium industry amounted to 6068, comprised of 2557 in mining and 831 in the mills, with the remainder described as general employees (1). The work force probably did not change significantly until recent months with the closure of Eldorado Nuclear and Madawaska Mines.

The estimated annual production capability from known Canadian deposits should increase from the present 7145 t U to 10 200 t U by 1991*. Annual fuel requirements for Canadian power plants are expected to grow from 110 t U in 1981 to between 2100 and 2400 t U in 1991 (1).

CURRENT TECHNOLOGY

The processing of uranium ores in Canada and throughout the world is essentially the same, with only slight variations from mill to mill. With the exception of the Eldorado Beaverlodge mill (recently closed) which processed the ore by alkaline leaching (2), all other plants in Canada have employed acid leaching (3-10). In Canada there are many uranium minerals identified (11) and the mineralogy of the particular uranium ore influences the process flowsheet (12) to an extent, but mineralogical differences can usually be handled by varying the operating parameters required for a particular stage. The economics of a particular flowsheet are dictated by the ore type and process technology used (13).

The general schematic flowsheet of a uranium processing plant is shown in Fig. 14. The ore is first crushed and ground in rod or ball mill circuits, or by semi-autogenous grinding. The crushed ore can be upgraded by electronic sorting in order to reject as much of the gangue material as possible before processing further. Preconcentration may be valid for application to certain ores. Leaching of the ground ore, at about 55% solids, is accomplished by atmospheric

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* Projection based on operating and committed production centres only; certain planned but as yet uncommitted expansions at operating production centres are excluded. Output from resources currently mineable at a uranium price of $135/kg U or less (1).

Table 13 - Canadian uranium ore processing plants

Plant	Location	Nominal daily capacity (tonnes)	Annual production rate (1) (tonnes U)	Total employees (mine, mill general)
A. Operating				
Agnew Lake Mines Limited	Agnew Lake, Ont.	n.a. (2)	195	79
Cluff Mining (Amok Ltd./SMDC)	Cluff Lake, Sask.	360 (3)	11	241
Denison Mines Limited - Denison Mill	Elliot Lake, Ont.	6440 (4)	1712	2
Eldorado Nuclear Limited	Eldorado, Sask.	1630 (5)	423 (6)	845 (11)
Gulf Minerals Canada Ltd.	Rabbit Lake, Sask.	1500 (7)	1960	320
Madawaska Mines Limited	Bancroft, Ont.	1360	235	381 (11)
Rio Algom Limited				
- Quirke Mill	Elliot Lake, Ont.	6350	1879	1404
- Panel Mill	Elliot Lake, Ont.	2990	730	771
B. Committed				
Key Lake Mining Corp.	Key Lake, Sask.	500-700 (8)	3100-4600	
Rio Algom Limited - Stanleigh Mill	Elliot Lake, Ont.	4540 (9)	-	
C. Planned and Possible (10)				
Brinco Limited	Makkovik, Nfld.	-	-	
Canada Wide Mines	Midwest Lake, Sask.	-	-	
Consolidated Rexpar Minerals & Chemicals Ltd	Birch Is., B.C.	-	-	
Norcen Energy Resource Ltd	Beaverdell, B.C.	-	-	
Rio Algom Limited - Milliken Mill	Elliot Lake, Ont.	-	-	

n.a. not applicable

- no data

(1) 1980 production for operating plants; planned rate for others.

(2) Using an underground and surface heap-leaching technique; ceased mining 1980 but continued with surface operations.

(3) A two-stage program, the first phase of which was completed in 1980.

(4) Increased to 13 610 t/d by mid-1981.

(5) Operated at about half nominal capacity in 1980.

(6) Includes 6 tonnes U from Cenex Limited.

(7) Joint venture with Uranerz Canada Limited; operating in excess of nominal capacity.

(8) Production expected by July 1983.

(9) Rehabilitation to be completed 1983-84.

(10) No firm decisions as to timing and size of operations.

(11) Mine and plant shut down in 1981.

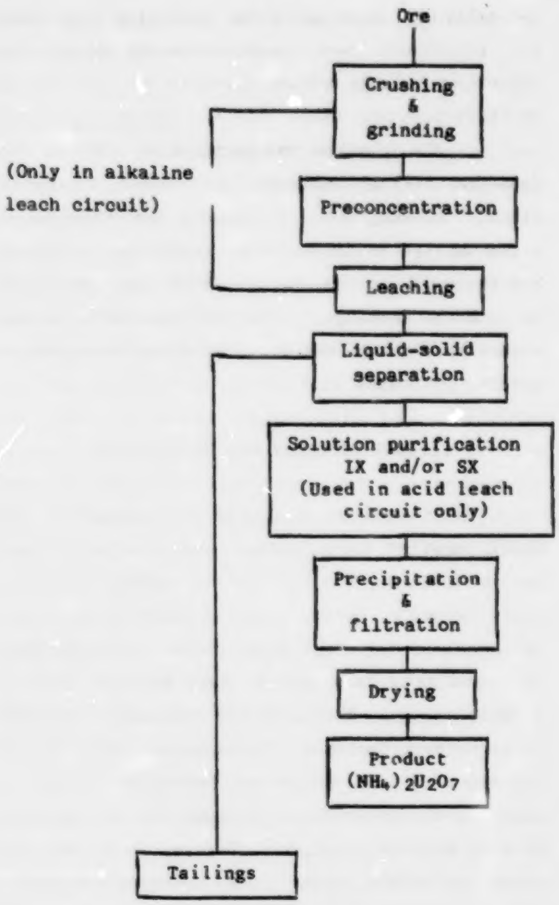
Ore
Crushing & grinding
(Only in alkaline leach circuit)
Preconcentration
Leaching
Liquid-solid separation
Solution purification IX and/or SX (Used in acid leach circuit only)
Precipitation & filtration
Drying
Product (NH₄)₂U₂O₇
Tailings

Fig. 14 - Schematic flowsheet for processing uranium ores

oxidation acid leaching in pachucas or air lift agitors (4,5,6,9,13); bacterial leaching as practised at Agnew Lake Mines on surface heaps (1,14); pressure oxidation leaching for complex ores of Saskatchewan (7) and alkaline oxidation leaching in pachucas was used at Eldorado Nuclear's Beaverlodge operation (2).

Following the liquid-solids separation by filtration, countercurrent decantation wash thickeners and clarification, the uranium-bearing filtrate is sent to purification and uranium recovery. In the case of the alkaline circuit, the uranium is precipitated with caustic, filtered, dried and purified by ion exchange (3,4,5,6,9,13) whereby the uranium is upgraded before the concentrated solution is precipitated with either ammonia or magnesia to recover either an ammonium or magnesium diuranate product. Other plants use solvent extraction to produce the upgraded and purified uranium product (7,8,10,13) which is similarly recovered as an ammonium or magnesium diuranate. This "yellow cake" product is filtered, dried and drummed for shipment to a buyer, or to the Eldorado Nuclear refinery in Port Hope for further refinement to products of UO_3 and UO_2 (15), which are subsequently clad in zircalloy tubing as fuel bundles for the Candu nuclear reactors (16,17).

Recovery of uranium in the overall process amounts to about 92%. Detailed description of plant flowsheets worldwide are given in several books by IAEA (12) and others (18,19).

The solid and liquid wastes are discharged to a tailing impoundment area as a slurry. The radium is precipitated by the addition of barium chloride (20,21), producing an effluent of less than 10 pCi/l Ra^{226}. The Madawaska mill was precipitating the radium prior to discharge to tailings, and storing in a separate holding area (9). Approximately 95-97% of the original radium in the ore reports in the solid tails, while 3-5% is in solution.

In addition to uranium, there have been byproducts produced over the years. The Port Radium mill of Eldorado Nuclear at one time produced a cobalt gravity concentrate for subsequent purification and use in cobalt therapy treatment. Rio Algom produced a thorium product from treatment of uranium barren solution, which was further refined in the UK (22). The Elliot Lake mills at one time produced a bulk rare earth concentrate also from treatment of uranium-barren liquors (23). These byproducts were sold in the USA. CANMET did considerable work in the early 1970s to improve the rare earth flowsheet (24). The complex high grade ores of Saskatchewan, notably Key Lake and Midwest Lake both contain nickel and arsenic (7,10) which eventually could be recovered as valuable byproducts. Although the thorium is not presently in demand, the longer term objective in the fuel cycle is to use thorium (25). At the present time even experimental quantities of thorium oxide for research on the fuel cycle are being imported into

Canada (25). In the long term Canada could probably recover a high purity thorium product from the Elliot Lake barren solutions. CANMET has done some work in that area (26). Rare earths are becoming increasingly useful in TV phosphors, flints, ceramics, glass, and are now in demand as catalytic converters in automobiles. With some effort and expenditure by Canadian industry, Canada could be a producer of all the rare earths plus yttrium by treating the Elliot Lake uranium barren solutions. At the present time the thorium, yttrium and rare earths are discarded in the barren solution sent to tailings.

PROBLEMS

Although the general processing practice at Canadian uranium mines has not varied much in the past 30 years, since it was developed by CANMET, nevertheless problems remain unresolved. The major concern is for the environment, health and safety of the workers, community and the surrounding environment (27,28,29). Because of the nature of the process, using H_2SO_4 leaching, 95% or greater of the radionuclides are in the tailings. Subsequently the Ra^{226} is slowly solubilized during air and bacteria oxidation and weathering of the tailings, resulting in the slow release of Ra^{226}. Such solubilized material must be treated by chemical precipitation with barium chloride to precipitate $RaBaSO_4$. Pyrite oxidation to sulphuric acid with the resultant leaching of other metals in the tailings also add to the tailings problem. Dusting of dry tailings, containing radon gas emanation, is often prevalent. Vegetation cover and stabilization of the tailings to prevent dusting and water seepage therefore become important. Thus there is a high cost entailed in present tailings management. Decommissioning of shut-down mine/mills also adds a very high cost (30).

Expanding development of the rare earth technology and, to some extent, that of uranium in other countries, could make it more difficult in the future for Canadian mills to remain competitive. China, for example, has the largest known reserves of rare earth minerals, and has the present technical capability for producing high purity, individual rare earth products (31). Also China has a large effort directed to the end use of these products (31).

The uranium industry, like most of the Canadian mining industry is located in remote areas. The manpower turnover is therefore large. A few of the mills, notably Cluff Lake of Amok, and the Gulf Rabbit Lake operation fly their crews in each shift change, This has resulted in a real reduction in turnover compared to other mines at similar latitudes (32).

CURRENT RESEARCH AND DEVELOPMENT

Current Research and Development on the development of improvements to conventional technology is being carried out by CANMET and to a small extent by custom research organizations such as Lakefield Research, Saskatchewan Research Council, and Ontario Research Foundation. Research is also being conducted at the following universities: Ottawa, Carleton, Saskatchewan, Waterloo and McMaster. A list of recent federally funded research projects is found in Appendix 1. Development of new, alternative technology is mainly centered at CANMET, where chloride based technology (33,34) is being studied This research has the major objective (35) of solubilizing the radionuclides for subsequent isolation in the mill, the disposal of an environmentally acceptable tailing, increased uranium recovery and increased recovery of byproducts.

GAPS AND CHALLENGES

As noted earlier, the general processing scheme for uranium has been practised for about 30 years with only incremental improvements. However, there are still areas where improvement in the unit process could result in increased recovery, decreased capital and operating costs, decreased energy requirements and increased byproduct recovery. Improvement in tailings technology must also be accomplished.

The following are areas within the uranium process flowsheet where there appear to be technology gaps, and where improvements to assist the process are required.

PRECONCENTRATION

The possibility of making a uranium concentrate by physical means (flotation, magnetic separation) for subsequent leaching and tailingS having an acceptably low level of radioactivity for safe, long-term storage should not be overlooked. However, for such an approach to be technically and economically feasible, the mineralogical characteristics of the ore would have to be such that 95% of the uranium could be concentrated in less than 15% of the weight. This is not the case for the Elliot Lake area where known concentration technology has reached the required recovery but in about double the desired weight (36,46).

LEACHING

Further work is required in the use of oxidants to improve the kinetics and completeness of leaching uranium (37). Also countercurrent leaching should be optimized. In the pressure leaching of uranium ores leach residues might be produced that possibly would be more stable to environmental weathering. Such a problem exists in the arsenical ores of Saskatchewan (7). During the leaching in sulphuric acid the radionuclides may initially be dissolved but appear to be precipitated on the gangue particles. Consequently, following the leach, only about 3-5% of the original Ra^{226} present in the ore is solubilized. Technology is required to promote solubilization the Ra^{226}, in the presence of a relatively large sulphate concentration, during leaching possibly by complexation.

LIQUID-SOLIDS SEPARATION

Reagent, energy and capital costs related to the liquid-solid separation stage are significant (13). Development of better surfactants and improved equipment could result in both reduced plant areas and equipment costs. Reduction of uranium soluble losses in the filter cake may also be possible. High capacity thickners and belt filters are examples of recent developments to optimize this stage of the process.

SOLUTION PURIFICATION

Conventional technology has employed ion exchange or solvent extraction for purification or direct precipitation (13). In the case of ion exchange, fixed bed or moving bed have been used, and now continuous ion exchange appears to be the better and more economical route (14,38,39). Two Canadian designs have been proposed, that of Himsley (40) which has enjoyed full-scale application and of CANMET (38,39) which has been tested successfully on a pilot plant scale. The Himsley column is now installed in plants throughout the world, one in Canada at Agnew Lake (14), and one is proposed for the Rio Algom Stanleigh mill (41). The NIMCIX continuous ion exchange column has been developed in South Africa by the National Institute of Metallurgy and is in use in several of that country's mills. Improvements to resin density and specificity will lead to improved yellowcake products at the mill site, and such a resin has been developed by Sentrachem in South Africa.

Solvent extraction has also undergone changes in operation, and equipment. Further improvements to the equipment and mode of operation can reduce the solvent loses due to entrainment in emulsions and crud (42,43). An improved understanding of the stripping characteristics that would result in a higher grade product at the mill site would be advantageous.

Some plants use a combination of ion exchange and solvent extraction. The Eluex (14) or Purlex processes (44) are examples. Any significant advances will be made by improving the IX and SX processes involved.

As mentioned earlier, the liquid-solids separation stage is costly. This step could be eliminated prior to uranium recovery, by treating the slurry directly for uranium recovery, then a substantial savings would result (45). Although Canada as well as other countries have made significant advances in solvent-in-pulp or ion exchange-in-pulp extraction techniques as a means for reducing the cost of recovering uranium from solution, much more can and should be done in this area (31).

PRECIPITATION, FILTRATION AND DRYING

There is often a problem in precipitation of the yellowcake and its subsequent filtration and drying because the scientific reasons for the problems that occur are not properly understood. Research in this area may lead to improved efficiency. Conventional precipitation of uranium for leach solutions is with ammonia, magnesia or caustic and peroxide. An improved understanding of the possibilities offered by a different route of precipitation to produce nuclear grade UO_3 or UO_2 at the mill site may result in reduced process costs. A uranium precipitate that could be dried at a lower temperature compared with conventional high temperature drying would reduce energy requirements.

BYPRODUCT RECOVERY

Bulk products of thorium or rare earths have been produced at one time but if Canadian industry is to compete with other countries in this area, more research is required to provide for high purity compounds. Additionally, there are other possible byproducts such as arsenic and nickel found in the northern Saskatchewan mills. Because these constituents are normally dissolved in the leaching process, some optimization of the leach conditions could maximize the dissolution of the byproducts. As they are essentially free to the process plant, the isolation for recovery of high purity compounds should be an incentive to develop technology for their recovery and possible end uses. However, any process to recover byproducts in uranium ore processing would have to take into account the need to eliminate radiation sources from the byproducts.

TAILINGS

Considerable research on migration of radionuclides and other constituents through uranium tailings has been done, but much remains to be accomplished in order to provide for optimum tailings site design. The type of tailings discharge method may be an important factor in diminishing seepage. The National Uranium Tailings Program, which has just been announced, will address the many aspects of uranium tailings and their management.

ANALYSES

More rapid analytical methods for application in uranium ore processing operations would be useful, particularly those for the determination of the radionuclides.

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APPENDIX - DIRECTORY OF FEDERALLY SUPPORTED RESEARCH IN UNIVERSITIES

1980-1981

Field investigation of radon gas emanation parameters from mine backfill containing uranium mill tailings; Archibald, J.F.; Wilson, E.G.; Mining Engineering; Queen's University, Kingston, Ont.; Energy, Mines and Resources; $8000

The extraction and recovery of uranium using open cell polyurethane foam sponge (OCPUFS); Gesser, H.D.; Chemistry; the University of Manitoba, Winnipeg, Man.; $9500

Hydrogeologic, hydrogeochemical and model studies of ground water contaminant migration from waste rock at an abandoned uranium mine near Bancroft, Ontario; Gillham, R.W.; Cherry, J.A.; Earth Sciences; University of Waterloo, Ont.; Atomic Energy Control Board; $12 154

Treatment of uranium mining effluents; Huck, P.M.; Systems Engineering; University of Regina, Sask.; Natural Sciences and Engineering Research Council; $7500

Organophosphorus compounds as extractants for uranium and thorium; Janzen, A.F.; Chemistry; University of Manitoba, Winnipeg, Man.; Natural Sciences and Engineering Research Council; $16 470

High temperature desulphurisation of gases using rare earth mineral concentrates; Kay, D.A.R.; Metallurgy/Materials Science; McMaster University, Hamilton, Ont.; Energy, Mines and Resources; $13 000

1. Evaluation of uranium distribution in lake water and sediment 2. Mineralogy, petrology and sulphide potential of layered gabbros in the Lynn Lake area 3. Uranium migration and deposition within the Athabasca formation; Parslow, G.R.; Geological Sciences; University of Regina, Sask.; Natural Sciences and Engineering Research Council; $9091

Minéralogie, pétrologie, géochimie et métallogénie de certaines occurrences d'uranium au Québec; Perrault, G.; Cameron-Schimann, M.; Technologie des minéraux; Ecole Polytechnique, Montréal, Qué.; Energie, Mines et Ressources; $4500

1979-1980

Application of impregnated fibres for sampling and removal of radium from uranium mine effluents; Bland, C.J.; Physics; University of Calgary, Alta.; Energy, Mines and Resources; $10 000

Hydrogeologic, hydrogeochemical and model studies of groundwater contaminant migration from waste rock at an abandoned uranium mine near Bancroft, Ontario. On behalf of AECB; Gillham, R.W.; Cherry, J.A.; Earth Sciences; University of Waterloo, Ont.; Supply and Services Canada; $55 335

Stable isotope (O,C,S,H) geochemistry of uranium mineralization; Morton, R.D.; Muchlenbachs, K.; Hattori, K.; Geology; University of Alberta, Edmonton, Alta.; Energy, Mines and Resources; $12 000

Minéralogie, pétrologie, géochimie et métallogenie de certaine occurrences d'uranium au Quebec; Perrault, G.; Technologie des mineraux; Ecole Polytechnique, Montréal, Qué.; Energie, Mines et Ressources; $3000

Conduct a quantitative assessment of polynuclear aromatic hydrocarbon levels in uranium mines. On behalf of EMR; Westaway, K.; Chemistry; Laurentian University of Sudbury, Ont.; Supply and Services Canada; $31 418

The extraction and recovery of uranium using open cell polyurethane foam sponge (OCPUFS); Gesser, H.D.; Chemistry; University of Manitoba, Winnipeg, Man.; Energy, Mines and Resources; $8000

The extraction and recovery of uranium using open cell polyurethane foam sponge (OCPUFS); Gesser, H.D.; Chemistry; University of Manitoba, Winnipeg, Man.; Energy, Mines and Resources; $8000

Organophosphorus compounds as extractants for uranium and thorium; Janzen, A.F.; Chemistry; University of Manitoba, Winnipeg, Man.; Natural Sciences and Engineering Research Council; $15 000

1. Evaluation of uranium distribution in lake water and sediment 2. Mineralogy, petrology and sulphide potential of layered gabbros in the Lynn Lake area 3. Uranium migration and deposition within the Athabasca formation; Parslow, G.R.; Geological Sciences; University of Regina, Sask.; Natural Sciences and Engineering Research Council; $8280

Conduct a quantitative assessment of polynuclear aromatic hydrocarbon levels in uranium mines; Westaway, K.; Chemistry; Laurentian University of Sudbury, Ont.; Energy, Mines and Resources; $31 418

1978-1979

Application of impregnated fibres for sampling and removal of radium from uranium mine effluents; Bland, C.J.; Physics; University of Calgary, Alta.; Energy, Mines and Resources; $10 000

Oxygen isotope fractionations and geothermometers among minerals of uranium deposits; Morton, R.D.; Muehlenbachs, K.; Geology; University of Alberta, Edmonton, Alta.; Energy, Mines and Resources; $8000

1. Evaluation of uranium distribution in lake water and sediment 2. Mineralogy, petrology and sulphide potential of layered gabbros in the Lynn Lake area 3. Uranium migration and deposition within the Athabasca formation; Parslow, G.R.; Geological Sciences; University of Regina, Sask.; Natural Sciences and Engineering Research Council; $8000

1977-1978

Investigation of radium content of effluent from uranium mines and methods for its removal; Bland, C.J.; Physics; University of Calgary, Alta.; Energy, Mines and Resources; $10 000

1. Evaluation of uranium distribution in lake water and sediment 2. Mineralogy, petrology and sulphide potential of layered gabbros in the Lynn Lake area 3. Uranium migration and deposition within the Athabasca formation; Parslow, G.R.; Geological Sciences; the of Regina, Sask.; National Research Council Canada; $7150

Distribution of uranium in minerals from selected ore environments; Zentilli, M.; Geology; Dalhousie University, Halifax, N.S.; Energy, Mines and Resources; $5000

1976-1977

1. Evaluation of uranium distribution in lake water and sediment 2. Mineralogy, petrology and sulphide potential of layered gabbros in the Lynn Lake area 3. Uranium migration and deposition within the Athabasca formation; Parslow, G.R.; Geological Sciences; University of Regina, Sask.; National Research Council Canada; $5000

Distribution of uranium in minerals from selected ore environments; Zentilli, M.; Geology; Dalhousie University, Halifax, N.S.; Energy, Mines and Resources; $9500

1975-1976

Mineralogical study of uranium-bearing materials from the Elliott Lake-Blind River area, Ontario; Berry, LG.; Thesis, M.J.; Geological Sciences; Queen's University, Kingston, Ont.; Energy, Mines and Resources; $7000

Oxygen isotope fractionations and geothermometers among minerals of uranium deposits; Morton, R.D.; Muehlenbochs, K.; Geology; University of Alberta, Edmonton, Alta.; Energy, Mines and Resources; $5500

1974-1975

Crystal structure studies of minerals, mineralogical and sedimentological study of uranium bearing minerals of the Blind River-Elliot Lake area, Ontario; Berry, L.G.; Geological Sciences; Queen's University, Kingston, Ont.; National Research Council of Canada; $7900

Mineralogical study of uranium-bearing materials from the Elliott Lake-Blind River area, Ontario; Berry, L.G.; Theis, M.J.; Geological Sciences; Queen's University, Kingston, Ont.; Energy, Mines and Resources; $6500

To study the sources of radon gas in Canadian uranium mines and their effects on mining methods and costs of mining; Thompkins, R.W.; Mining

Engineering; Queen's University, Kingston, Ont.; National Research Council of Canada; $5000

1973-1974

To study the sources of radon gas in Canadian uranium mines and their effects on mining methods and costs of mining; Thompkins, R.W.; Mining Engineering; Queen's University, Kingston, Ont.; National Research Council of Canada; $5000

1972-1973

To continue studying the physical parameters affecting radon emanation in uranium mines with particular attention to geological nonconformities and rock stresses; Thompkins, R.W.; Mining engineering; Queen's University, Kingston, Ont.; National Research Council of Canada; $5750

To study the sources of radon gas in Canadian uranium mines and their effects on mining methods and costs of mining; Thompkins, R.W.; Mining Engineering; Queen's University, Kingston, Ont.; National Research Council of Canada; $5000

1971-1972

To study the sources of radon gas in Canadian uranium mines and their effects on mining methods and costs of mining; Thompkins, R.W.; Mining Engineering; Queen's University, Kingston, Ont.; National Research Council of Canada; $4000

Radon emanation rates in uranium mines; Thompkins, R.W.; Mining Engineering; Queen's University, Kingston, Ont.; Mines Branch (EMR); $4500

Electron microprobe study of rare-earth-bearing minerals from alkalic rock complexes; Watkinson, D.H; Geology; Carleton University, Ottawa, Ont.; Geological Survey of Canada (EMR); $1255

COAL

GENERAL DESCRIPTION OF INDUSTRY

In 1980, Canadian coal production was approximately 36 Mt. Sixteen million tonnes consisted of Alberta subbituminous coals and Saskatchewan lignites produced in captive operations adjacent to thermal power generating plants where coal preparation amounted to screening and crushing the coal prior to pulverizing for combustion purposes. The balance of the national production (20 Mt) was bituminous coal of metallurgical and thermal quality, destined for export and interprovincial markets. Essentially all of this production was processed (washed) coal, the output of ten (10) mine-site washeries in Alberta, British Columbia and Nova Scotia. The total installed capacity of these operations was approximately 6000 t/h run-of-mine coal.

All ranks of coal considered, Alberta was the major producer followed in order by British Columbia, Nova Scotia, Saskatchewan and New Brunswick. Of the estimated $1100 million total value of output, $867 million was the result of export, primarily to Japan, from British Columbia and Alberta bituminous coking coal producers.

Two metallurgical coal mines ceased operation in Alberta in 1980 because of economic and marketing difficulties related in part to decline in reserves. These shut-downs reduced installed washery capacity by about 700 t/h. However, new installations now under construction are expected to increase coal washing capability approximately 1.5 times to about 9000 t/h by 1985. Turn-of-the-century requirements are projected at 3-5 times the 1980 capacity or 18 000 to 30 000 t/h installed.

CURRENT TECHNOLOGY

The purpose of coal preparation is to enhance the usability and value of the as-mined coal by reduction of the ash, sulphur and moisture contents as well as control of the coal size distribution. Developments in the preparation technology now in place have come about both as a response to the customer requirements based on existing utilization technologies and also as a direct result of some of the increased use of mechanized mining techniques and the development of stringent environmental protection regulations.

The basic operations of coal washing involve breaking and screening the raw coal, separation for removal of undesirable constituents, dewatering, thermal drying, water treatment, recovery of usable process materials where applicable, and waste disposal. In Canada, approximately 88% of the output from bituminous coal mines was washed using separation processes based on differences in densities between the coal and ash components. Process selection is generally based on a consideration of cost-effectiveness, given the product specifications, raw coal characteristics, tonnage and known process limitations. The most consistent limitation of density-related processes is the rapid drop in efficiency as particle size decreases below 0.5 mm. For this reason processes which effect separation according to differences in surface properties of fine particles, e.g., froth flotation, are normally used in the finer size range. Froth flotation was employed on the 12% balance of mine output.

COAL WASHING PROCESSES

A distinction is made between coarse and fine coal processes with the dividing line usually set at 0.5 mm. Coarse coal washers are all based on separations due to density differences and operate in the particle size range 127-0.5 mm. They include the jig, heavy medium vessel, heavy medium cyclone and water cyclone and account for about 68% of the combined annual throughput of Canadian washeries (Table 14). Fine coal washers which include the water cyclone, concentrating

Table 14 - Installed capacities for cleaning processes used in Canadian washeries (1980)

Cleaning process	Capacities Mt/h Western Canada	Capacities Mt/h Maritimes	Capacities Mt/h Total	Per cent Western Canada	Per cent Maritimes	Per cent Total
Heavy medium cyclone*	2124	476	2600	34.5	7.7	42.2
Water only cyclone*	1141	27	1168	18.5	0.4	18.9
Heavy medium vessel*	736	-	736	12.0	-	12.0
Jig*	718	27	745	11.7	0.4	12.1
Flotation	560	204	764	9.1	3.3	12.4
Table*	145	-	145	2.4	-	2.4
TOTAL	5424	734	6158	88.2	11.8	100.0

* Process based on density differences.

table and flotation are used for the balance of production. Each of these processes is described below.

COARSE COAL CLEANING

For coal in the size range 127 mm to 10 mm two types of coal cleaning equipment are dominant.

Jig

A bed of raw coal in a water bath is stratified by a regular series of pulsations such that the lighter coal particles move to the top and the heavier ash particles sink to the bottom as shown in Fig. 15.

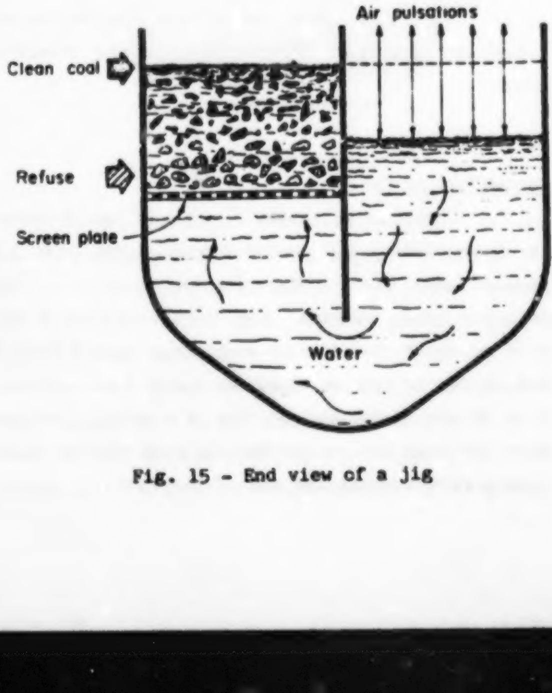

Fig. 15 - End view of a jig

Jigs are generally low cost units offering a medium to high efficiency and are used primarily for cleaning thermal coals. The Canadian installed capacity is in four plants and accounts for 10% ($47 million) of our cleaned coal product. The waste losses from these units are estimated at 0.33% of product ($156 000).

Heavy Medium Vessel

Raw coal is immersed in a "heavy medium" and cleaned strictly by differences in specific gravity. The specific gravity of the medium -- usually a suspension of finely divided magnetite in water -- is adjusted such that the high gravity material (ash) sinks and is removed as refuse. The clean coal product floats off as shown in Fig. 16. The specific gravity at which the separation occurs can be adjusted by controlling the concentration of the magnetite in the slurry. This is a high efficiency system when used with large sized coal. The Canadian installed capacity of 735 t/h is devoted to metallurgical coal cleaning; it accounts for 12% of our installed capacity with a product value of about $170 million. The waste losses from these units amount to approximately 1.6% of product or $2.8 million per year.

INTERMEDIATE SIZE COAL CLEANING

For coal in the size range 35 mm to 0.5 mm, the efficiency of both the jig and the heavy medium vessel deteriorates. In this size range

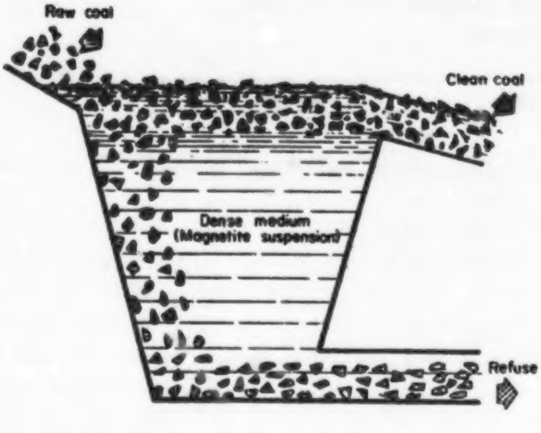

Fig. 16 - Simplified view of a heavy medium vessel

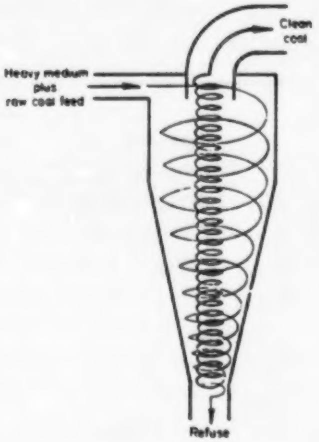

Fig. 17 - The heavy medium cyclone

the heavy-medium-cyclone and the water-only-cyclone are widely used.

Heavy Medium Cyclone

These units are derived from the heavy medium vessel; the gravity separation between coal and ash is accelerated by the centrifugal forces of the cyclone. The central vortex carries the low gravity coal particles through the cyclone and the high gravity ash is discharged at the apex of the cone as refuse. This gives a much more efficient separation than the heavy medium vessel alone. Individual cyclones have throughputs up to 100 t/h.

The Canadian installed capacity in heavy medium cyclones is spread across both eastern and western coalfields and amounts to 2400 t/h and is responsible for the major part of coal preparation in Canada. The estimated annual value of the product streams is $560 million; losses in the refuse streams account for 0.23% of the product and are valued at $1.3 million per year.

Water-only-Cyclones

These are based on the same design principles as the heavy medium cyclones. The separation of coal from ash is controlled by variations in the design parameters for the cyclone and by the velocity of the water within the cyclone. They make a less efficient separation than the heavy-medium cyclone but have the advantage that no provision need be made for the recovery of the heavy-medium material.

Water cyclones account for 19% of our installed coal cleaning capacity with a cumulative throughput of 1150 t/h. The value of coal cleaned in this way is approximately $215 million; waste stream coal losses are valued at $12 million.

FINE COAL CLEANING

For coal in the size ranges below 3 mm, the efficiency of the cyclone and jig systems is poor. It becomes necessary to use a separation system based on coal properties other than specific gravity if the ash is to be removed to give a marketable quality coal.

In this size range the concentrating table and the froth flotation cell are commonly used.

Concentrating Table

This is a ribbed, inclined table which is given horizontal reciprocation. The coal is carried down the incline over the ribs by a low velocity water stream. The combination of downward movement due to the wash water and sideways motion due to the oscillations leads to a separation of the clean coal at the base of the incline with the high ash refuse moving with the reciprocating ribs towards the end of the table.

These units are generally of low cost and are more efficient than water cyclones when required to clean fine coal. Canadian capacity is limited to one plant with a 145 t/h installation. The output of this plant is valued at $34 million per annum; the waste stream has a high coal content (20%) with losses valued at $6 million.

Froth Flotation Cell

This cell uses the surface characteristics of fine coal particles which allow them to become attached to air bubbles passing through a coal slurry. Ash particles, on the other hand, do not possess this surface characteristic and do not attach themselves to the air bubbles. A clean coal "froth" therefore develops at the top of the cell with an ash refuse being discharged from the base. The technique is restricted to coal sizes below 0.5 mm; the units are expensive and generally of low capacity and performance can be sensitive to the chemical characteristics of the coal surface. An illustration of a froth flotation cell is shown below.

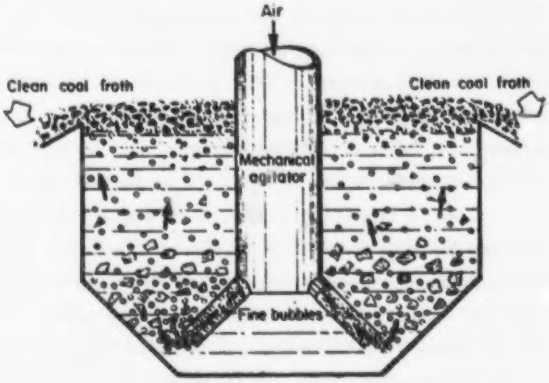

Fig. 18 - Froth flotation cell

The Canadian installed capacity of 732 t/h is spread across 5 plants and accounts for 12% of our coal cleaning capacity. The clean product is valued at about $170 million and the system losses at $24 million.

The preponderance of highly mechanized coal mining operations leads to a run-of-mine product which is not efficiently cleaned by the coarse coal cleaning technologies. In addition, the geological development of the coal seams of Western Canada has given rise to a highly fissured coal which breaks down rapidly during mining and transportation. The technologies for intermediate and fine coal cleaning therefore dominate the industry; this situation is unlikely to change.

The performance data for the coal preparation industry for 1980 indicate that the industry as a whole suffered losses of 4% or $50 million as a result of process shortcoming. Nearly all of these losses ($45 million) occurred in the cleaning of fine coal sizes below 0.5 mm. This inefficiency loss relates only to the performance of the preparation equipment and is incremental to mining and transportation losses.

MECHANICAL DEWATERING, DRYING AND WATER TREATMENT

Water is removed from all washery products - from the coarsest to the finest clean coal and reject. Water removal from the clean coal enhances its value and reduces shipping costs, improves handling and prevents freezing. Removal from the rejects, although it entails largely non-recoverable costs, improves handling and stability and prevents environmental contamination following disposal. The recovered water must meet stringent quality criteria to permit recycling to the plant or disposal to the environment. The importance of water treatment as an integral operation in coal preparation is often underestimated since it apparently does not add directly to product value. However, the costs of not treating the water would add to the costs of operation as manifested by reduced plant capacity, increased losses of saleable coal, increased losses of magnetite in heavy medium processes, increased costs of maintenance and water supply and environmental contamination.

Mechanical Dewatering

The difficulty and costs of dewatering increase proportionally to decreases in particle size. The separation processes involve the use of stationary/vibrating screens, thickening cyclones, vacuum/pressure filters, centrifuges and sedimentation vessels. In an analagous manner to the individual coal-ash separation processes, both the coal-water and the ash-water separation processes are most efficient when applied to discrete ranges of solid particle sizes.

Screens are low-cost, low capacity systems which can be applied equally well to large sized coal or reject streams. With particle sizes above 10 mm a feed coal stream can be efficiently reduced to a surface moisture content below 10%. Screens usually perform multiple functions as sizing, rinsing and dewatering equipment, and in many instances form the first stage in a multistage water removal process. In Canadian coal preparation plants, screens are used to give a final clean product at a total production rate of 320 000 t/a.

Centrifuges are used for dewatering coal in the size range 25 mm-0.5 mm as screen bowl systems; the less efficient solid bowl centrifuge is dominantly used in fine refuse dewatering. In both applications they are low-capacity, high-cost units giving rise to a product with a moisture content between 10 and 12%. Canadian capacity is approximately 1080 t/h and is responsible for dewatering approximately 45% of our clean coal production.

Filters are low capacity, high cost units that are usually used to treat the product froth from a froth flotation cell and are restricted to coal particle sizes below 0.5 mm. The vacuum disc systems shown in Fig. 19 are the most common. In these units, suction is applied to one side of a cloth or screen surface to draw the solids and water towards the surface; the filtration surface retains the solids but allows the water to pass through.

Filters account for about 45% of our final stage clean coal dewatering capacity.

Thermal Drying

Thermal dryers are used only in Western Canada for final drying of metallurgical coals. In most cases the feed to the dryer is a mixture of the output from dewatering screens, centrifuges and vacuum filters blended to give a stream with a moisture content from 13 to 15%. The dried product contains approximately 6% moisture.

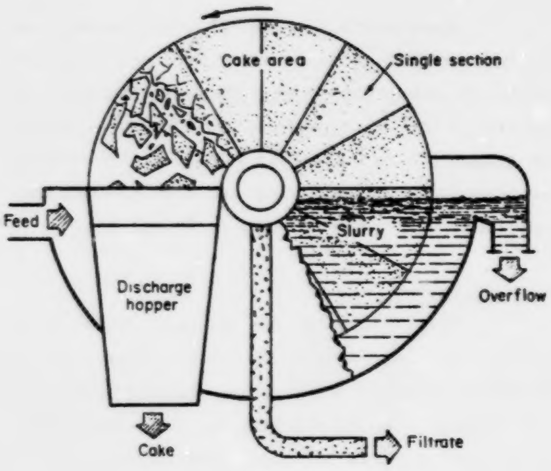

Fig. 19 - Rotary disc filter

The installed Canadian capacity is capable of handling 3000 t/h.

Water Treatment

The process of water treatment in coal washeries involves clarification of a portion of the total plant water by sedimentation of the finest solid particles in tailings thickeners and/or ponds, and thickening of the sedimented layer to the extent that the solids can be effectively dewatered in filters or centrifuges and/or pumped to the impoundment area. Since the sedimentation rate is proportional to particle size and the area required to effect the separation of the solid and the liquid increases as particle size decreases, chemical coagulants and flocculants are added to the thickener feed stream to produce aggregates of fine particles that settle more rapidly. The clean water is recycled to the washing circuits.

Tailings thickeners are utilized in eight washeries. The feed solids usually consist of flotation tailings and/or classifier cyclone and screen undersizes finer than 0.25-0.15 mm and at concentrations from 3-8% by weight. Thickener areas vary from 186 to 4470 m^2 and total 12 600 m^2 for the eight plants. Hourly throughput totals 14 745 t of effluent consisting of 345 t of solids and 14 200 m^3 of water. This throughput amounts to approximately 2 Mt of solids and 84 Mm^3 of water per year. The costs of water clarification/tailings thickening for all washeries amount to $2.6 million per year of which chemical reagent costs account for $1.5 million (60%). As a charge against clean coal value, the cost generally varies between $0.09 and $0.11/t of product but may be as high as $0.22. From the perspective of water clarification for environmental protection and water conservation the costs vary from $0.02 to $0.06/m^3. From the point of view of the thickening aspect of the operation, costs vary between $0.80 and $2.00/t of tailings with an average of $1.38 for the eight plants. However, this cost relates only to the first step of fine tailings treatment: an additional expenditure varying between an estimated $0.06 and $0.30/t, depending on choice of subsequent dewatering method(s), would be needed to achieve a technically and environmentally acceptable product for disposal.

Total costs of the solid-liquid separation processes involving removal of water from clean product and water clarification/tailings thickening are conservatively estimated at $20.5 million/a. The greatest single expenditure amounting to $13.5 million/a arises from the drying operation. In all, removal of water from the clean product by centrifugation, filtration and drying requires an average expenditure of over $0.90/t for a total of $17.9 million/a.

This outline has focussed on the solid-solid and solid-liquid separation processes that are used in coal preparation. It is emphasized, however, that technical shortcomings which add to the cost of processing are aggravated by mechanical deficiencies arising from poor design, improper use and/or inappropriate applications. Some of these can be alleviated through adequate maintenance and attention of skilled operators. Coal preparation technology therefore includes various elements of mechanical, materials, and systems engineering in addition to the processing aspect itself.

PROBLEMS

Many of the problems associated with coal preparation are shared with other countries. However, there are two unique problems to Canadian coals. Coal friability common in Western Canadian coals and finely disseminated pyritic sulphur in Eastern Canadian coals.

COAL FRIABILITY

Western Canadian coals are known for their friability which leads to the run-of-mine coal containing up to 60% smaller than 0.5 mm. Tectonic movements are the major cause of extensive fracturing of coal in the mountain districts. Mining methods and handling and processing contribute to the generation of fines. However, the amount of fines generated during mining, handling and preparation depends significantly on the degree of natural fissuration.

The relative friability of different coals is of great importance in preparation because the greater the proportion of fines the greater the total preparation costs. The preparation cost per ton of feed is really a function of the number of particles per ton. Thus, preparation costs increase rapidly as the mean particle size becomes smaller. For example, preparation of coal less than 5 mm in diam costs four times more in capital investment and five times more in operating expenses than coarse coal above 0.5 mm. New preparation plants are usually designed at an early stage of the development of a property before the start-up of mining activities. Underestimating the amount of fines leads to overloading of fines cleaning, dewatering, drying and water recovery circuits and causes operating problems and losses of fine coal to the discard pile. A recent performance evaluation study of Canadian washeries showed that yield losses in the fine circuits were approximately 10 times those occurring in the coarse coal circuits.

SULPHUR REDUCTION

Increasingly more stringent sulphur emission regulations to alleviate acid rain problems are forcing the coal industry to search for improved means to reduce sulphur content. Maritime coals are known for their high sulphur content (2-6%). Iron pyrite is responsible for 30-70% of the total sulphur and in most cases is finely disseminated. Crushing and grinding are required to liberate pyritic sulphur. The resulting decrease in the particle size of the coal produces an exponential increase in preparation cost.

Sulphur reduction in coal can be achieved by partial removal of pyritic sulphur depending on pyrite particle size and distribution. Organic sulphur can only be removed through chemical cleaning which is a "future technology" with much higher cost than the currently mature technologies. Density separators can remove pyrite particles to 50-70 μm, however at smaller sizes, density separation is inadequate. Two stage froth flotation has been shown to remove pyrite particles finer than 50 μm on a pilot plant scale; the process requires demonstration on an industrial scale.

Successful reduction of sulphur through R & D in the Maritimes coal would increase use of coal locally and increase chances for export, particularly to Europe.

CURRENT RESEARCH AND DEVELOPMENT

CANMET

For about 20 years the Coal Research Laboratory in Edmonton has been in the forefront of Canadian coal preparation research. R & D efforts are directed at resolving problems that occur in coarse and fine coal preparation. The program is carried out at four levels: laboratory and bench scale, mini plant, pilot plant and mobile plants. It includes studies of coal degradation, coal structure characterization, water treatment, flotation of partially oxidized coal, dewatering, online instrumentation, process control and optimization and modelling.

The Mineral Sciences Laboratories in Ottawa is involved in sulphur removal from high sulphur coal, chemical comminution, simulated processing of plants and dry separation.

NATIONAL RESEARCH COUNCIL OF CANADA

The NRC Chemical Engineering Laboratory at Ottawa for over a decade has been developing and improving an oil agglomeration process. Work currently involves recovery and dewatering of fine coal from washeries tailings and is being undertaken using a mobile demonstration plant.

ATLANTIC CANADA FACILITIES

The Research and Productivity Council of New Brunswick and the Technical University of Nova Scotia are active in investigating coal cleaning options for sulphur reduction including chemical cleaning. Research is restricted to the bench scale.

ONTARIO HYDRO RESEARCH

Research is concentrated on fine coal cleaning for sulphur reduction. A small pilot plant facility is used to conduct R & D.

COAL MINING RESEARCH CENTRE - ALBERTA

The CMRC programs includes moisture reduction, dry cleaning, washing of low rank coal, waste disposal and utilization. Current research plans are restricted to the drycleaning of Alberta sub-bituminous coals.

COAL AND MINERAL PROCESSING CENTRE - BRITISH COLUMBIA

Laboratory and pilot plant facilities are used for teaching and research. Areas of research include coal handling, oxidized coals, cleaning and dewatering of fines, tailings dewatering and dry separation.

GAPS AND CHALLENGES

The outlined summary of the various coal preparation processes and problems points to the fact that a number of areas exist where improvements will result in substantial economic and environmental benefits as manifested in greater production (recovery and grade), less costly and environmentally more acceptable operations.

These improvements can be classified into two main categories: those for processes cleaning coarse sizes and those for cleaning fine coal. The former relates mainly to improvements of the efficiency of separation equipment which could be achieved by operational optimization and/or mechanical optimization. Operational optimization can be obtained as a result of the application or development of better monitors and controls promoted by the "electronic revolution" and mechanical optimization by better measurements and controls of the functions of the equipment itself. Optimization by improved control of the heavy medium bath density has resulted in sharper separations; equipping cyclones with controlled underflow spigots and changing the feed inlet geometry again gave sharper separations; better controls on jig operations resulted in the Batac Jig. A long list of similar examples could be cited; however the underlying fact is that these improvements have mainly been adopted as a result of developments in areas other than coal preparation, and a great deal more needs to be done to bring coal preparation up to the technological forefront reached by many other industries.

Traditional fines treatment in coal preparation plants was to deslime the coarse product on screens and send the fines to disposal areas. Lack of developments in this area was due to the difficulty of characterizing the different properties of fine coal and fine reject on which their separation could be based. This is still the case. Additionally, the interactions between the different components and additives to the system as well as their behaviour in the separation equipment are difficult to determine involving, as they do, the chemical and physical nature of the coal as well as the engineering design of the system equipment.

Much of the research in this area up to now has been typified by two categories of approach, highly empirical, i.e., case specific, or purely theoretical dealing with idealized conditions which do not exist in nature. The former has the inherent lack of generalization and extrapolation; each case is dealt with on its own merits and only temporary half-measure solutions can be achieved. The purely theoretical approach lacks realism, and by itself fails to solve industrial problems.

Research in coal fines recovery has been directed to flotation with much less effort devoted to such processes as spherical oil agglomeration, high intensity magnetic separation, selective flocculation, chemical and bacterial leaching and electrostatic separation. Efforts have been equally distributed, in this area, between development of chemical additives (flocculants, flotation reagents), process efficiency enhancement (design of circuitry of separation), and minor improvements in equipment. The development of chemical reagents has had very little emphasis and has been mainly carried out at chemical companies and is subject to proprietary restriction on the information developed. Selection of the proper reagents and optimization of their application has been carried out in numerous agencies as well as in preparation plants but mainly on a trial-and-error basis.

In summary, coarse coal preparation is a more established technology, fines preparation stands to improve the most through research. Areas of great potential benefit for the Canadian coal industry are coal characterization in relation to coal preparation and utilization, process control and plant automation, dewatering and water recovery, coal preparation in relation to utilization, and sulphur reduction.

The emphasis in the future will focus on the design of fully automated preparation plants where the modular and distributed control system concept is employed. This requires development and design of adaptive control schemes and development of probabilistic models for all preparation systems. Deep cleaning will be applied to produce high quality products of metallurgical coal for export and low ash and sulphur coal for thermal use to compete on the international market. Deep cleaning will require finer liberation sizes and new techniques for transporting coal slurries to minimize the need for dewatering and drying and to reduce dust problems associated with handling fine coal. This will support coal as an economically justifiable energy source and will also make it easier to handle and use with minimum environmental hazards. Emphasis in future R & D will involve linking coal preparation to new technology development to meet requirements for both combusion and conversion. Reduction of losses in fine coal can be achieved through closely controlled and optimized operations resulting in high recovery and less waste disposal problems. In general, efficient and improved coal preparation technology will reduce the cost of transporting coal to distant markets, both to Central canada and overseas.

OPINION POLL

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